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SEPTEMBER, 1915

Bulletin of the American Institute of Mining Engineers

PUBLISHED MONTHLY

SECTION I

	PAGE
Advisory Board for United States Navy..	iii
Library Service Bureau	iv
Nominations for Officers	v
Personal	v
Employment	vii
Library	ix
Membership	xi
Institute Committees	xxiii
Officers and Directors	xxx

SECTION II.—PAPERS AND DISCUSSIONS

Important Topping Plants of California. By Arthur F. L. Bell 1769

The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba. By James F. Kemp. 1801

Tramming and Hoisting at Copper Queen Mine. By Gerald F. G. Sherman 1837

The Pacific Coast Iron Situation. By Charles Colcock Jones 1887

The Furbero Oil Field, Mexico. By E. DeGolyer 1899

Manufacture and Tests of Silica Brick for the Byproduct Coke Oven. By Kenneth Seaver 1913

Mine Pumping. By Charles Legrand 1929

Conveyor-Belt Calculating Chart. By J. D. Mooney and D. L. Darnell. 1937

Ventilation of the Copper Queen Mines. By Charles A. Mitke. 1941

The Formation of the Oxidized Ores of Zinc from the Sulphide. By Y. Tsenshan Wang. 1959

The Stresses in the Mine Roof. By R. Dawson Hall. 2013

Kick vs. Rittinger: An Experimental Investigation in Rock Crushing, Performed at Purdue University. By Arthur O. Gates 2023

Standardizing Rock Crushing Tests. By Myron K. Rodgers. 2053

Discussion of the Paper of Roswell H. Johnson, The Rôle and Fate of the Connate Water in Oil and Gas Sands. By C. W. Washburne 2057

Rotary Kilns for Desulphurization and Agglomeration. By Samuel E. Doak. 2061

The Mexican Oil Fields. By L. G. Huntley. 2067

The Cost of Maintaining Production in California Oil Fields. By M. E. Lombardi . . . 2107

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PROPOSAL FOR MEMBERSHIP

Mr. ______________________________
(Name in Full)

Occupation ______________________________

Address ______________________________

is hereby proposed by the undersigned, as a _______________

____

of the American Institute of Mining Engineers.

____

____

____

Signatures of three Members or Associates.

Place of birth ____________________ *Year of birth* ________

Education, general and technical, when, where and how acquired, with degrees, if any.

Dates	

Record of experience. Briefly, the past and present employment, with names of employers, companies and associates. (Proper names, names of companies, etc., should be written without abbreviations.)

Dates	

Present position ______________________________

____

______________________________***Signature.***

Dated ______________________________***191***

EXTRACTS FROM THE CONSTITUTION.

ARTICLE II.—MEMBERS.

SEC. 1. The membership of the Institute shall comprise four classes, namely: 1. Members; 2. Honorary Members; 3. Associates; 4. Junior Members. * * *

SEC. 2. The following classes of persons shall be eligible for membership in the Institute, namely: as Members, all professional mining engineers, geologists, metallurgists, or chemists, and all persons actively engaged in mining and metallurgical engineering, geology, or chemistry; as Associates, all persons desirous of being connected with the Institute who in the opinion of the Board of Directors are suitable.

As Junior Members, all students in good standing in engineering schools who have not taken their degrees and who are nominated by at least two of their instructors. * * *

Every candidate for election as a Member, Associate, or Junior Member must be proposed for election by at least three Members or Associates, must be approved by the Committee on Membership, as prescribed in the By-Laws, and must be elected by the Board of Directors.

BULLETIN OF THE AMERICAN INSTITUTE OF MINING ENGINEERS

PUBLISHED MONTHLY

No. 105 SEPTEMBER 1915

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Entered as Second Class matter January 28, 1914, at the Post Office at York, Pennsylvania, under the Act of March 3, 1879.

ADVISORY BOARD FOR UNITED STATES NAVY

The members of the Institute have probably seen in the daily papers notices of the plans of the Secretary of the Navy to form an Advisory Board to assist the Government to make available the latest inventive genius of the country. Secretary Daniels has appointed Mr. Thomas A. Edison Chairman of this Advisory Board and has officially invited the Institute to secure the selection of two of its members to serve on this Board. The Secretary in his letter states:

> "I feel that the work your Institute has done has been such as to give it the right to be in a way officially represented, and the Navy Department desires in this way to testify to its own appreciation of the splendid work for our country that your Institute has done. I have the emphatic approval of Mr. Edison, and he agrees entirely with me that your Institute should be represented in this way and that no better method of getting the kind of men we need could be devised."

The invitation of the Secretary of the Navy has been officially accepted and steps have been taken to nominate the two members. Since the Constitution provides for no meeting of the Board of Directors until September, and as Secretary Daniels urged early action, the matter was discussed at a special meeting of the Executive Committee of the Institute, at which were present as many members of the Board of Directors as were in New York City at the time. Invitations were also extended to other members of the Institute who happened to be in the vicinity. At this meeting it was decided to submit the matter by a special letter-ballot to the Board of Directors and in doing so to suggest or nominate five members of the Institute, accompanying this nomination with the statement that the Directors might vote for two of the members so nominated or for any other members of the Institute not so nominated. It was voted at this meeting that it was the sense of those present that in view of the fact that this committee was an advisory one the type of men to be chosen should be men of administrative or executive ability, capable of passing upon the value of inventions and of obtaining facilities for securing technical advice on questions which might be presented, rather than that the men so chosen should be of distinctive prominence as inventors. No nominations were made at the meeting, but every one was advised to communicate by letter-ballot with the Secretary of the Institute, it being understood that the five names having the highest number of votes should then be transmitted as suggestions to the Board of Directors for their letter-ballot, members of the Board being free to select their own candidates. Nearly all of the 24 members of the Board of Directors have voted, and the Secretary, acting under instructions, has maintained absolute privacy as to the expression of each Director. A meeting of the Executive Committee will be called at an early date, when the two members of the Institute receiving the highest number of votes will be announced and transmitted to the Secretary of the Navy for appointment. The officers of the Institute do not feel free to make any announcement of the names of the members selected until they have received the consent of Secretary Daniels to do so.

LIBRARY SERVICE BUREAU

The Library Board of the United Engineering Society is desirous of bringing closer to you the facilities and usefulness of your splendid Library. To that end, they have inaugurated a Service Bureau whose duty it shall be to maintain such a staff of expert searchers and translators as to insure prompt and thoroughly efficient service to those of our members whose absence from New York makes it inconvenient for them to pesronally consult the books and periodicals on our shelves.

The Service Bureau is also arranging to keep such members as may desire it, posted on the current publications of any engineering subject in which they may be specially interested.

The Charter of the United Engineering Society does not permit the Library to carry on any work of a commercial character and, therefore, the prices charged members for this service have to be kept down to a merely self-supporting basis.

All the work done by the Library Service Bureau is strictly confidential.

The combined libraries of the American Institute of Mining Engineers, the American Society of Mechanical Engineers, the American Institute of Electrical Engineers and the United Engineering Society, which are now conjointly administered by a Library Board of the last-named Society, have on their shelves over 60,000 books on technical subjects, and they receive currently over 1,000 engineering or scientific periodicals.

For those members who are personally able to use the library a bibliography on any desired engineering subject will, on request, be prepared by the Bureau, and with adequate prior notice, the volumes relating thereto will be set apart for personal perusal.

We desire that all members of the Institute shall become better acquainted with the splendid library facilities they own, and with the advantages which can be secured by those who avail themselves of the aid furnished by its Service Bureau.

NOMINATIONS FOR OFFICERS

The co-operation of the members of the Institute is earnestly sought by the Committee on Nominations, recently appointed by the Board of Directors, in its work of formulating a ticket for officers and places falling vacant in the Board of Directors in February, 1916. The Committee must present the official ticket of nominations to the Board of Directors at its October meeting. Members are, therefore, urged to send promptly their nominations for officers and Directors in order that the Committee may have an opportunity to give them full consideration.

The officers to be elected at the annual meeting in February, 1916, are: One officer, known as Director and President. Two officers, known as Director and Vice-President. Five officers, known as Director.

The officers of the Institute whose terms expire are as follows:

President, William L. Saunders (not eligible for re-election).

Past President, Charles F. Rand.

Vice-Presidents, Thomas H. Leggett, District 0; Fred W. Denton, District 4.

Directors, John W. Finch, District 7; John H. Janeway, District 0; Edward P. Mathewson, District 5; Joseph W. Richards, District 2; George D. Barron, District 0.

Communications should be sent to the Chairman of the Committee on Nominations, Crocker Bldg., San Francisco, Cal.

COMMITTEE ON NOMINATIONS: F. W. BRADLEY, *Chairman.*

LOUIS S. CATES, JAMES F. KEMP, FRANK M. SMITH,
R. C. GEMMELL, FRANK A. ROSS, ARTHUR THACHER.

PERSONAL

(Members are urged to send in for this column any notes of interest concerning themselves or their fellow-members.)

Members and guests who registered at Institute headquarters during the period July 10 to Aug. 10, 1915:

William H. Rettie, Yonkers, N. Y.
A. J. Eveland, Pachuca, Mexico.
Chang Fu Wang, Shanghai, China.
F. H. Fovargue, Terlingua, Tex.
W. W. Elmer, Tuttletown, Cal.
Paul S. King, Wilmington, Del.
E. E. R. Tratman, Chicago, Ill.
Courtenay Dekalb, Tucson, Ariz.
Emil Gathmann, Baltimore, Md.

Henry S. Drinker, E. M., LL. D., has just completed 10 years' service as President of Lehigh University. A notable gathering of alumni, with their wives and daughters, numbering more than 1,000 persons, met in South Bethlehem on June 5. In accordance with the request of Dr. Drinker, no formal celebration was held, but, nevertheless, a spontaneous and enthusiastic expression of appreciation and affection was voiced by alumni, students and faculty, testifying to the regard felt for Dr. Drinker and extolling the progress made by the University under his guidance.

Thomas T. Read, for several years Associate Editor of the *Mining and Scientific Press*, has accepted a position with the Engels Copper Co., Engels, Cal.

M. S. Brandt has severed his connection with the Seneca Consolidated Gold Mines Co., Seneca, Cal., and has resumed work with the White Eagle Mining & Smelting Co., Sunshine, Colo. He has also opened an office for general practice.

Milo W. Krejci, Metallurgist of the Boston & Montana Reduction Works of the Anaconda Copper Mining Co., has been made Assistant Superintendent.

Earl S. Bardwell, who was General Foreman of the concentrator, has been appointed Metallurgist of the Boston & Montana Reduction works.

Theodore P. Holt has been appointed Superintendent of the Tintic Milling Co., Silver City, Utah.

Benjamin C. Rogers has accepted a position in the Department of Geology of the Detroit Copper Co., Morenci, Ariz.

Arthur Clark Terrill, who as field representative of the New York State Museum, collected and arranged the large mining exhibit of the Empire State at the Panama-Pacific International Exposition, has been appointed head of the department of Mining Engineering at the University of Kansas, Lawrence, with the title of Professor of Mining and Ore Dressing, beginning Sept. 1, 1915.

William B. Phillips, formerly Director of the Bureau of Economic Geology and Technology at the University of Texas, Austin, will assume the Presidency of the Colorado State School of Mines, Golden, Colo., on Sept. 1, 1915.

S. M. Marshall is now manager of the turbine, condenser, and pump departments of the Southwark Foundry & Machine Co., Philadelphia, Pa.

Irving R. Gard is now consulting engineer for the Columbia River Coal Dock Co., North Portland, Ore.

Luther B. Eames has accepted the position of Mill Superintendent of the Hollinger Gold Mines, Ltd., Timmins, Ont.

Robert Ammon has been appointed Assistant Mill Superintendent for the American Zinc Co., Mascot, Tenn.

Charles E. Locke was a guest of alumni of the Massachusetts Institute of Technology at a recent informal dinner in Denver, and gave an illustrated lecture on the new home of that Institute.

Lawrence Arthur Callaway was granted the degree of Doctor of Science by Harvard University at the recent commencement exercises. The subject of his thesis was The Constitution of Iron-Copper Mattes.

Carl F. Schaber has been appointed division engineer for the Tennessee Coal, Iron & Railroad Co., Bessemer, Ala.

Drury McM. Phillips has severed his connection with the Consumer's Lignite Co., Hoyt, Tex., and has accepted a position with the engineering division of the refinery department of the Texas Company at its Port Arthur Refinery.

Walter E. Burlingame announces that the business of E. E. Burlingame & Co., assayers and chemists, will be conducted under his name at 1736–38 Lawrence St., Denver, Colo.

Milton A. Allen has been appointed Manager of the Pearsite Company, Inc., Cannel City, Ky.

Harry P. Sweeny is General Manager of the Canadian Atlantic Coal Co., Ltd., Port Malcolm, Nova Scotia.

John Orth Liebig is now employed by the Balbach Smelting & Refining Co., Newark, N. J.

C. Edwin Nighman has been appointed shift boss at the Leonard mine of the Anaconda Copper Mining Co., Butte, Mont.

ENGINEERS AVAILABLE

(Under this heading will be published notes sent to the Secretary of the Institute by members or other persons introduced by members.)

Member, technical graduate, aged 36. Experienced copper metallurgist and mining geologist. Familiar also, through 15 years' broad practical experience, with mining, cyanidation, concentration (including oil flotation), metallurgical accounting, and mine and plant management. Best references. No. 237.

Member, graduate engineer with 10 years' experience in malleable, gray iron, and steel, as chemist, metallographist, and metallurgist. Researches and investigations by laboratory methods and under works conditions. No objection to some routine. No. 243.

Member, technical graduate, broad experience in coal, oil, and clay. Will go anywhere as mining geologist after Sept. 1, 1915. No. 244.

Member, technical graduate, aged 30, married, with over 10 years' experience as chemist, engineer on construction, superintendent and general superintendent of bituminous coal and coke plants. Speaks Spanish. References. No. 245.

Member, open for engagement as mine safety inspector or as superintendent of bituminous coal mine. No. 246.

Member, graduate mining engineer, with five years' experience in coal mining as mining engineer, construction engineer, pit boss, and superintendent, open for engagement. Best references. No. 247.

Member, aged 39 years, 20 years' underground experience gained in Canada, Washington, California, Nevada, Colorado, Mexico and Central America. Speaks Spanish. Assayer and surveyor. Good on maps. Years of experience as foreman and superintendent. Up-to-date on mining costs and methods. Practical. Wants position as general mine foreman or assistant to superintendent. At present in Canada. No. 248.

Member, technical graduate, aged 32, married, 12 years' practical experience in mining and metallurgy of gold and copper, desires change. Now with company operating continuous decantation cyanide process. Available Sept. 1. No. 249.

Research chemist, technical graduate, aged 30. Seven years' varied experience as metallurgical chemist in several industrial concerns engaged in the manufacture of special alloys, both ferrous and non-ferrous. For the past 3½ years chief chemist in charge of two busy laboratories for present employers. Best reasons for desiring change. Excellent references from present and past employers. Open for engagement after Oct. 1. Correspondence arranging for personal interview invited. No. 250.

Member, aged 49, married, 25 years' experience in hydraulic mining: Speciality, opening up new properties. Not a college graduate. At present general manager of mining company. Open for engagement after Nov. 1, as manager or superintendent. No. 251.

Member, technical graduate, 8 years' experience, three in metal and five in coal mine engineering. Have made a study of ventilation and efficiency. 32 years old, single and temperate. Desire a position as mine or ventilation engineer, superintendent or inspector of mines, will go anywhere. Can report immediately. No. 252.

Member, technical graduate, wide experience in western United States, Mexico, South America and the Orient. Organization and management of properties. Economic and metallurgical investigation. Estimates design and construction of plants. No. 253.

ADVANTAGES TO MEMBERS OF OUR ADVERTISING SECTION

It is our intention to make the advertising section of the *Bulletin* as comprehensive as possible, so that members can turn to it for complete information as to where to buy the best mining and metallurgical supplies at the lowest prices. No firms or supplies of dubious reputation are advertised in the *Bulletin*. If members, in buying any material, will refer to the advertisement of that material in the *Bulletin*, or if they do not find it advertised in the *Bulletin*, will, when ordering, ask the manufacturers why the goods were not advertised in the Institute publication, it will be of material benefit to all of us, because the chief reason why advertisers are unable to trace direct results is because purchasers do not mention the *Bulletin* when writing. Finally, if members, when in need of anything whatsoever, will write to the Institute headquarters, an effort will be made to send them a complete list of manufacturers able to supply the desired materials of good quality. The furnishing of these lists will be a benefit to our Advertising Department, as well as to the members. We need scarcely call attention to the advantages obtained by the members from the money which advertising brings in, because all this money is returned directly to the members in one form or another.

LIBRARY

AMERICAN INSTITUTE OF ELECTRICAL ENGINEERS
AMERICAN SOCIETY OF MECHANICAL ENGINEERS
AMERICAN INSTITUTE OF MINING ENGINEERS
UNITED ENGINEERING SOCIETY

WILLIAM P. CUTTER, Librarian

The Library of the above-named Societies is open from 9 A.M. to 10 P.M. on all week-days, except holidays, from September 1 to June 30, and from 9 A.M. to 6 P.M. during July and August. The Library contains about 55,000 volumes, including sets of technical periodicals and the publications of scientific and technical societies.

Members of the Institute, with few exceptions, are forced to spend a portion of their time in localities isolated from sources of information. To these the Library can render valuable service through correspondence; letters requesting information will receive special attention. The Library is prepared to furnish references and copies of articles on mining and metallurgical subjects; to determine the existence of mining maps, and to furnish general information as to the geology and mineral resources of all countries.

All communications should be made as definite as possible so that the information received may be what is desired and not include collateral matter which may not be of interest. The time spent in searching for such collateral matter will be saved, and the information will be sent more promptly and in more usable shape.

The members of the Institute can be of service to the Library by forwarding copies of mining reports, maps privately issued, and similar material, which will be classified, indexed, and made available to other members. Suggestions for additions to the Library, either by purchase or personal solicitation as gifts, will be welcomed. It is hoped that members while in the city will use the Library freely, and assurance is given that most careful service will be rendered to them.

LIBRARY ACCESSIONS

PARTIAL LIST CLASSIFIED BY SUBJECTS

Mining, Metallurgy, and Chemistry

ATLANTIC CITY GOLD MINING DISTRICT, FREMONT COUNTY. (Bull. 7, Wyoming State Geologist.) Cheyenne, 1914.
CHEMISTRY OF PETROLEUM AND ITS SUBSTITUTES. A Practical Handbook. By C. K. Tinkler and F. Challenger. London, 1915.
CHEMICAL ENGINEERING NOTES ON GRINDING, SIFTING, SEPARATING AND TRANSPORTING SOLIDS. By. J. W. Hinchley. London, 1914.
HANDBUCH DER MINERALCHEMIE. Bd. 11, pt. 7. By C. Doelter. Dresden, 1915.
KUPFER. By W. Borchers. Halle, 1915.
COKE-OVEN ACCIDENTS IN THE UNITED STATES DURING THE CALENDAR YEARS 1913, 1914. (Technical Paper 118, U. S. Bureau of Mines.) Washington, 1915.
PRACTICAL COAL MINING. Ed. 5. By George L. Kerr. London, 1914.
METHODS OF PREVENTING AND LIMITING EXPLOSIONS IN COAL MINES. (Technical Paper 84, U. S. Bureau of Mines.) Washington, 1915.

OCCURRENCE OF EXPLOSIVE GASES IN COAL MINES. (Bull. 72, U. S. Bureau of Mines.) Washington, 1915.
PERMISSIBLE EXPLOSIVES, TESTED PRIOR TO MARCH 1, 1915. (Technical Paper 100, U. S. Bureau of Mines.) Washington, 1915.
WYOMING. MINING LAWS, FEDERAL AND STATE, 1913. (Bull. 6, Wyoming State Geologist.) Cheyenne, 1913.
THE TINTIC MINING DISTRICT OF UTAH, prepared for the Tintic meeting of the Utah Section of the American Institute of Mining Engineers, July 15, 1915, with map. (Gift of Chief Consolidated Copper Co.)

Geology and Mineral Resources

CLAY AND SHALE DEPOSITS OF THE PROVINCE OF QUEBEC, PRELIMINARY REPORT. (Memoir 64, Canada Department of Mines.) Ottawa, 1915.
COAL FIELDS OF BRITISH COLUMBIA. (Memoir 69, Canada Department of Mines.) Ottawa, 1915.
COAL FIELDS OF MANITOBA, SASKATCHEWAN, ALBERTA, AND EASTERN BRITISH COLUMBIA. (Memoir 53, Canada Department of Mines.) Ottawa, 1914.
LES MINERAIS DE FER DE LA PROVINCE DE QUÉBEC, GISEMENTS ET UTILISATION. Quebec, 1915.
MARBLE AND MARBLE WORKING. By W. G. Renwick. London, 1909.
BASIN AND GREYBULL OIL AND GAS FIELDS, BIGHORN COUNTY, WYOMING. (Bull. 10, Wyoming State Geologist.) Cheyenne, 1915.
PROSPECTIVE OIL FIELDS AT UPTON, WESTON COUNTY, ETC. (Bull. 5, Wyoming State Geologist.) Cheyenne, 1913.
SALT CREEK OIL FIELD, NATRONA COUNTY. (Bull. 8, Wyoming State Geologist.) Cheyenne, 1914.
LITTLE BUFFALO BASIN OIL AND GAS FIELD. (Bull. 11, Wyoming State Geologist.) Cheyenne, 1915.
GEOLOGY AND ORE DEPOSITS OF COPPER MOUNTAIN AND KASAAN PENINSULA, ALASKA. (Professional Paper 87, U. S. Geological Survey.) Washington, 1915.

General

CATALOGUE OF TECHNICAL PERIODICALS IN LIBRARIES IN THE CITY OF NEW YORK AND VICINITY. Compiled by A. J. Gates. New York, 1915.
EXPERIENCES IN EFFICIENCY. By B. A. Franklin. New York, 1915.
SCIENCE AND PRACTICE OF MANAGEMENT. By A. H. Church. New York, 1914.
GUIDE BOOK OF THE WESTERN UNITED STATES. PART B—THE OVERLAND ROUTE. (Bull. 612, U. S. Geological Survey.) Washington, 1915.
INTERNATIONAL MINING MANUAL, 1915. Denver, 1915.
POOR'S MANUAL OF PUBLIC UTILITIES, 1915. New York, 1915.
MODERN PUMPING AND HYDRAULIC MACHINERY. By Edward Butler. London, 1913.
STRAIGHT LINE ENGINEERING DIAGRAMS. By Manifold and Poole. San Francisco, n.d.
STRUCTURAL ENGINEERING. By J. E. Kirkham. Chicago-London, 1914.
STRUCTURAL ENGINEERS' HANDBOOK. By M. S. Ketchum. New York, 1915.

Trade Catalogues

BESSEMER GAS ENGINE CO., Grove City, Pa. Bessemer Monthly. July, 1915.
TROPENAS CONVERTER CO., New York, N. Y. Steel Castings. Nos. 4, 5. Feb., July, 1915.

MEMBERSHIP

New Members

The following list comprises the names of those persons who became members during the period July 10 to Aug. 10, 1915:

Members

Albertson, Maurice Merton, Mine Surveyor, Mining Corporation of Canada, Cobalt, Ont., Canada.
Armstrong, George Chandler......Mgr., Indian Refining Co., Lawrenceville, Ill.
Barber, Max Hayden, Min. Engr..........Supt., Crosby Mine, Nashwauk, Minn.
Barneson, John..................Merchant, 310 Sansome St., San Francisco, Cal.
Bensel, John A., Cons. Engr...................111 Broadway, New York, N. Y.
Borgeson, Anshelm Cyrus, Min. Engr.........Shenango Mine, Chisholm, Minn.
Brooks, Charles Peter, Min. Engr.......228 Dooly Bldg., Salt Lake City, Utah.
Brown, Frank Harold, Cons. Min. Engr., Federal Co., Inc., 50 Broad St., New York, N. Y.
Clark, Douglas..Kellogg, Idaho.
Cresson, Benjamin Franklin, Jr., Chief Engr., Dept. of Commerce and Navigation, Jersey City, N. J
Crossfield, J. T. K., Min. Engr., Surveyor, Chile Exploration Co., Chuquicamata, Chile.
Daly, William Barry, Genl. Supt. of Mines, Anaconda Copper Mining Co., 508 Hennessy Bldg., Butte, Mont.
Dupuis, Oliver Camille, Mining..Mandarin Mines Corpn., Yellow Jacket, Idaho.
Edwards, Charles Lewis Taylor, Blast Furnace Assistant, Bethlehem Steel Works, So. Bethlehem, Pa.
Graff, Wilbur W., Supt., North Lake District, Cleveland-Cliffs Iron Co., Ihpeming, Mich.
Guggenheim, Daniel, Pres., American Smelting & Refining Co., 120 Broadway, New York, N. Y.
Hammon, Wendell C., Mining..............433 California St., San Francisco, Cal.
Handley, Harvey Lockhart......... Genl. Supt., Yankee Fuel Co., Yankee, N. M.
Harley, George Townsend, Experimental work, Arizona Copper Co., Ltd., Morenci, Ariz.
Harrington, Daniel, Min. Engr..........U. S. Bureau of Mines, Pittsburgh, Pa.
Herman, John, Assayer and Chem...........514 S. Los Angeles, Los Angeles, Cal.
Hilby, George Robert, Engineering and contracting................Salinas, Cal.
Hodge, John E., Min. Engr., Vice-Pres., E. J. Longyear Co., Minneapolis, Minn.
Hodgkinson, Harold Howe, Min. Engr., New Jersey Zinc Co., Franklin Furnace, N. J.
Holbrook, E. A.....Asst. Prof. of Mining Engineering, Univ. of Illinois, Urbana, Ill.
Koch, Arthur William, Mech. Draftsman, Care American Zinc Co. of Tenn., Mascot, Tenn.
Kwong, Wu.......Director, Tung Hsing Colliery, Men-tou-kou, via Peking, China.
Laroque, Fulgence, Min. Engr.; Shift Boss, Durban Roodepoort Deep Gold Min. Co., Ltd., Durban Deep, Roodepoort, Transvaal, So. Af.
MacIsaac, Frederick J., Contractor.............18 E. 41st St., New York, N. Y.
Marsh, Austin Gerry, Met., Concentrator Dept., Nevada Consolidated Copper Co., McGill, Nev.
Nelson, Carl Nelson, Min. Engr.; Mine Foreman, New York & Honduras Rosario Min. Co., San Juancito, Honduras.
Pettis, Edson S., Met.; Supt., California Ore Testing Co. Plant, San Francisco, Cal.
Reed, Avery H., Min. Engr........Cons. Min. Engr., Reed & Wilson, Marion, Ky.
Roberts, Hugh M., Min. Geol.......710 Security Bank Bldg., Minneapolis, Minn.
Roberts, William Paxton, Mining; Asst. Mine Surveyor, Witherbee, Sherman & Co., Mineville, Essex Co., N. Y.
Salmon, Herbert S., Chief Mine Inspector, Ore Mines, Care Tennessee Coal, Iron & R. R. Co., Bessemer, Ala.
Slaughter, Ben Gabriel, Chief Engr., Canadian Copper Co., Copper Cliff, Ont., Canada.

STACK, FRANK LAWRENCE, Min. Engr.............1547 Euclid Ave., Berkeley, Cal.
STRACHAN, FRANCIS J., Supt., Concentrator, Cananea Cons. Copper Co., Cananea, Son., Mexico,
TALLY, ROBERT E.........Supt. of Mines, United Verde Copper Co., Jerome, Ariz.
THOMPSON, LESTER S., Min. Engr........1373 Washington Ave., New York, N. Y.
TOBELMANN, HENRY ADOLPH, Met.; Chem., New Cornelia Copper Co., Ajo, Pima Co., Ariz.
WANG, CHENG-FU........................3 Quinsan Gardens, Shanghai, China.

Associate Members

ACKERMAN, ERNEST ROBINSON, Pres., Lawrence Portland Cement Co., Plainfield, N. J.
ALBIN, LELAND DAY, Asst. Gen. Sales Mgr., Ingersoll-Rand Co., 11 Broadway, New York, N. Y.
CONNABLE, FRANK L., Vice-Pres., E. I. du Pont de Nemours Powder Co., Wilmington, Del.
GODDARD, PETER LOUIS, Smelter Foreman, Mount Elliott, Ltd., Selwyn, Cloncurry, Queensland, Australia.
LEITER, JOSEPH, Coal Operator.................Dupont Circle, Washington, D. C.
PATTERSON JOHN H..............Pres., National Cash Register Co., Dayton, Ohio.

Junior Members

CROSTON, JOHN J., Met........................242 Church St., Grass Valley, Cal.
DAMMANN, ARTHUR CHESTER, Min. Engr..........................Fairview, Nev.
DAVID, JOSEPH, Student......................233 College Ave., Houghton, Mich.
EMMERSON, ERIC ROY..............................Port Arthur, Ont., Canada.
HICKS, FRANK VYVYAN, Engr................Care General Delivery, Bisbee, Ariz.
KAANTA, HENRY W., Assayer and Chem., Cariman Min. & Mill. Co., Cardinal, Colo.
ROHRER, WALTER ELMER, Asst. Engr.....Central Coal & Coke Co., Pittsburg, Kan.
SCHUYLER, ARENT HENRY.................404 Riverside Drive, New York, N. Y.
TAYLOR, WILLIAM WALTER, Engr., Canadian Copper Co., Copper Cliff, Ont., Canada.
WRIGHT, FREDERICK SYLVESTER, Asst. Chem., Boston & Montana Smelter, Great Falls, Mont.

Change of Status—Junior Member to Member

PRATT, ALLYNE FRANCIS........................727 E. 43d St. N., Portland, Ore.

CANDIDATES FOR MEMBERSHIP

APPLICATION FOR MEMBERSHIP.—The Institute desires to extend its privileges to every person to whom it can be of service. On the other hand, it is not desirable that persons should be admitted to membership in classes for which they are not qualified. Members of the Institute can be of great service if they will make a practice of glancing through the list of applicants and promptly notifying the Committee on Membership, or the Secretary of the Institute, of any persons whom they think should not be classified in accordance with the list given.

The following persons have been proposed during the period July 10 to Aug. 10, 1915, for election as members of the Institute. Their names are published for the information of Members and Associates, from whom the Committee on Membership earnestly invites confidential communications, favorable or unfavorable, concerning these candidates. A sufficient period (varying in the discretion of the Committee, according to the residence of the candidate) will be allowed for the reception of such communications, before any action upon these names by the Committee. After the lapse of this period, the Committee will recommend action by the Board of Directors, which has the power of final election.

Members

Wesley Hudson Beck, Ellsworth, Pa.
Proposed by W. A. Luce, W. W. Keefer, Samuel A. Taylor.
Born 1867, Rome, N. Y. Prepared self for college. 1886–88, Civil Engineering, Lehigh Univ. 1888–90, Land Surveying, California. 1890–91, Transitman, Rome, Watertown and Ogdensburg R. R. 1891–1903, Transitman, Asst. Engr. Maintenance of Way, Resident Engr. Construction, New York Central & Hudson River R. R. 1903–10, Asst. Engr., Pittsburgh Coal Co.
Present position: Asst. Chief Engr., Ellsworth Collieries Co.

James A. Block, Salt Lake City, Utah.
Proposed by J. M. Callow, Ernest Gayford, Walker J. Boudwin.
Born 1895, Salt Lake City, Utah. 1907–11, Salt Lake High School. 1911–14, University of Utah. 1915, Research work, U. S. Bureau of Mines, Salt Lake City. 1911, Draftsman, General Engrg. Co., 1913–15, Transitman and Draftsman, L. C. Stubbins, C. P. Brooks. 1914, Concrete inspector and engineer, Phoenix Construction Co.
Present position: Met. for J. M. Callow.

William M. Bradley, Salt Lake City, Utah.
Proposed by W. Fitch, J. R. Finlay, H. P. Henderson.
Born 1861, Elkhorn, Wis. 1883, Univ. of Wisconsin; LL. B. Owner of or interested in Centennial Eureka Min. Co.; Daly Judge Min. Co.; Silver King Cons. Min. Co.; Chief Cons. Min. Co.; Bennett Min. Co.; Centennial Emma Min. Co.; Good Hope Placer Min. Co., and other mines in the U. S. and Mexico.
Present position: Counsel for mining, smelting, railroad and other large corporate interests.

Grant David Bradshaw, Johnstown, Pa.
Proposed by D. M. Stackhouse, George E. Thackray, M. G. Moore.
Born 1884, Chicago, Ill. 1896–1901, Lewis Institute, Chicago, Ill. 1901–04, Univ. of Mich.; B. S. in M. E. 1904–09, Steam expert, draftsman, Secy. to Chief Engr., Foreman of Rod Mills, Joliet Works, Illinois Steel Co. 1909, Asst. Supt. Billet Mill, Indiana Steel Co., Gary, Ind. 1910, Consulting Engr., Chicago, Ill.
Present position: 1910 to date, Steam Engineer, Cambria Steel Co.

Ralph G. Bulkley, Montezuma, Colo.
Proposed by Frank Bulkley, Richard A. Parker, S. A. Ionides.
Born 1892, Aspen, Colo. 1909, Grad., East Denver High School. 1911, Phillips Academy, Andover, Mass. 1914, Yale University, Sheffield Scientific School; Ph. B. 1914, Miner, Golden Cycle Mine, Victor, Colo. 1914, Concentrating mill, Portland Cons. Gold Min. Co., Victor, Colo. 1915, Min. Engr., Crested Butte Coal Co. and Crested Butte Anthracite Min. Co., Crested Butte, Colo.
Present position: Supt. of Concentrating Mill, Montezuma Mines and Mills Co.

Stanley C. Bullock, Collahuasi, Chile.
Proposed by J. H. Ivey, Frederick Hellmann, Henry Hay.
Born 1881, Aylsham, Norfolk, England. 1889–94, Private school. 1904–08, Lancing College, Shoreham, Sussex, England. 1898–1902, Pupil, Fraser & Chalmers Works, Erith, erecting, turning, fitting, etc. Four years Engrg. course, Woolwich Technical College. 1902–05, Drawing office; mining, milling, smelting machinery and plants; details and design. Three years, assaying course, Woolwich Technical College. 1905–07, Draftsman and Asst. Engr., Bewick, Moreing & Co., London. Assaying and metallurgy course, Birkbeck College. 1907–09, Asst. Engr. and draftsman, Bewick, Moreing & Co., Australia. Australian Smelting Corporation. Surveying course, Melbourne School of Mines. 1909–11, Constructional Engr. Junction North Mine, Broken Hill (N.S.W.). Designed and erected new lead concentration and flotation plants; full charge during Chief Engineer's absence. 1911, Reporting on lead-zinc property, Norway. Sampled mines and advised on reconstruction of mill. 1911–12, Asst. Engr., Spassky Smelter Siberia; had charge of blast furnaces, converters and refinery. Learnt Russian and Kirghese. 1912–13, Reporting for various London firms. 1913–15, Mine and Mill Supt., Société des Mine de Cuivre, Collahuasi, La Grande, Chile.
Present position: Genl. Mgr., Poderosa Min. Co.

Albert Cameron Burrage, Jr., Boston, Mass.
Proposed by Huntington Adams, Fred Hellmann, E. A. C. Smith.
Born 1887, Boston, Mass. 1904–08, Harvard; B. A. 1908–10, C. S. Bradley & Sons, metallurgical work. 1910–12, Copper Extraction Co., metallurgical work.
Present position: 1912 to date, Director, Chile Copper Co.

J. H. Champion, Leadville, Colo.
Proposed by David G. Miller, W. W. Davis, William J. Cox.
Born 1856, Cornwall, England. 1874, Attended South Kensington class of Arts and Sciences; circumstances alone prevented pursuance of these studies. 1876, Contracted with Brendon Hills mine, Somersetshire, to produce iron ore. 1877–81, Contract work for Terrible Min. Co., Georgetown, Colo. 1881–86, Foreman, Small Hopes Co., Leadville, Colo. 1886–1890, Operating lease on Morning Star mine, Leadville, Colo. 1890–95, General contract work—sinking shafts and driving tunnels, Leadville, Colo. 1895–96, Foreman, Forest Queen mine, Leadville, Colo. 1896–1901, Foreman, The Yak Min., Mill. & Tunnel Co., Leadville, Colo.
Present position: 1901 to date, Supt., Yak Min., Mill. & Tunnel Co.

William Andrews Clark, Jr., Butte, Mont.
Proposed by W. A. Clark, Horace V. Winchell, Fred T. Greene.
Born 1877, Deer Lodge, Mont. 1899, University of Virginia; B. L.
Present position: Pres., Elm Orlu Min. Co., and Timber Butte Mill. Co.

Grant H. Dowell, Douglas, Ariz.
Proposed by C. Legrand, P. P. Butler, G. D. Van Arsdale.
Born 1866, Lexington, Ill. Up to 1884, common and high school, Kansas. 1892, Course, Kansas City Business Univ. 1897–1904, Chief Clerk and Asst. to Mgr., Cons. Kansas City Smelt. & Refg. Co., El Paso, Tex. 1904–10, Met. Accountant and ore buyer, Copper Queen Cons. Min. Co., Douglas, Ariz. 1910–12, Supt., Old Dominion Copper Min. & Smelt. Co., Globe, Ariz.
Present position: 1912 to date, Asst. Genl. Mgr., Copper Queen Cons. Min. Co.

Ensor R. Dunsford, Quarry, N. S., Canada.
Proposed by H. J. Brown, Laurance 1. Neale, F. C. Fearing.
Born 1887, Lindsay, Ont., Canada. Grammar and high schools, Columbus, O. 1911, Ohio State Univ.; E. M. 1911, Asst. Engr., Smithers Creek Dist., Sunday Creek Coal Co., Longacre, W. Va. 1912, Min. Engr., Darby Coal Min. Co., Darbyville, Va. 1912–15, Salesman and demonstrator, Myers-Whaley Co., Knoxville, Tenn.
Present position: Asst. Supt., Victoria Gypsum Min. & Mfg. Co., Ltd.

G. Marshall Gillette, Jenkins, Ky.
Proposed by Newell G. Alford, Howard N. Eavenson, Edward O'Toole.
Born 1879, Cumberland, Md. Prior to 1897, common school, Philadelphia, Pa., and Cumberland, Md. 1897–99, Princeton Univ.; B. S. 1903–13, Supt., Penn. Div.; 1913, Asst. Genl. Supt., Elk Horn Div.; 1913–14, Mgr., Millers Creek Div., Consolidation Coal Co., Van Lear, Ky.
Present position: Mgr., Elk Horn Div., Consolidation Coal Co.

Dorsey Hager, Tulsa, Okla.
Proposed by H. A. Wheeler, P. N. Moore, Arthur Thacher.
Born 1887, St. Paul, Minn. 1902–05, St. Louis Manual Training School. 1905–06 and 1907–09, Washington Univ., St. Louis, Mo. 1909–10, Columbia Univ., New York; specialized in Geology. Studied science and engineering at Washington Univ. 1906–07, Engrg. dept., St. Louis Street Railways. 1907, Katherine mine (lead), Fredericktown, Mo. 1909, Coal prospecting, Hamilton, Ia. 1910–12, Southern Pacific oil dept. (K. T. & O. Co.), San Joaquin Valley oil fields, Cal. Two and one half years, engrg. and geol. work, Cal. oil fields. Eight months, geol. work, Okla. oil fields. Also examinations made in N. M. and La.
Present position: Cons. work, Petroleum Geol. and Engr.

Oliver Hall, Coniston, Ont., Canada.
Proposed by C. V. Corless, John F. Robertson, R. W. Schultz.
Born 1879, Washington, Ont., Canada. 1903, Grad., McGill Univ.; E. M.; 1904, M. Sc. 1900–04, Mining, railway work (Grand Trunk) and Allis-Chalmers, Chicago (summers only). 1904–05, Engr., West Canadian Collieries, Blairmore, Alta., Canada.
Present position: 1905 to date, Mines Supt., Mond Nickel Co.

Ralph Lucius Healy, Thane, Alaska.
Proposed by B. L. Thane, Angus Mackay, B. B. Nieding.
Born 1887, Marshall, Minn. 1905, High school, Red Lake Falls, Minn. 1906–08, Univ. of Minnesota, College Elec. Engrg. 1909–11, Univ. of North Dakota, College Min. Engrg.; E. M. 1911–12, Vesuvius Mines Co., Bohemia, Ore. 1912–13, Northern Ontario Exploration Co., Juneau, Alaska.
Present position: 1913 to date, Asst. Engr., Perserverance Mine, Alaska Gastineau Min. Co.

Herbert Benton Henegar, Mascot, Tenn.
Proposed by Robert Ammon, M. H. Newman, H. K. Sherry.
Born 1891, Knoxville, Tenn. 1898–1908, Knoxville City School. Diploma from High School. 1908–12, Univ. of Tennessee; B. S. 1907–12, Summer vacations, surveying. 1912–15, drill runner, mine and mill sampler, warehouse man, surface foreman, American Zinc Co. of Tenn.
Present position: Asst. Mine Foreman, American Zinc Co. of Tenn.

Edwin Gilbert Izod, Johannesburg, So. Africa.
Proposed by W. L. Saunders, George A. Howells, R. S. Carter.
Born 1876, Southsea, England. 1885–93, Vickerys Army and Navy College, Southsea, England. 1893–99, Pupil, Maudsley Sons & Field, Lambeth, London (marine engineers). 1899–1901, University College, London; Junior and Senior Technical Course. 1901–10, Willians & Robinson, Ltd., Rugby, from experimental asst. to Asst. Managing Director.
Present position: 1910 to date, Cons. Elec. and Mech. Engr., Central Min. & Investment Corpn., Ltd., and Rand Mines.

William Ralph Knapenberger, Mascot, Tenn.
Proposed by A. W. Koch, J. N. Houser, Robert Ammon.
Born 1890, Brunswick, Mo. 1907, Brunswick Public School. 1908, St. Charles Military Academy. 1913, Missouri School of Mines; B. S. 1912, Asst. Chem., National Malleable Castings Co., Toledo, O. 1913–14, Investigator, Anaconda Copper Min. Co., B. & M. Reduction Wks., Great Falls, Mont.
Present position: Assistant in Mill, American Zinc Co.

Grover B. Lantz, San Francisco, Cal.
Proposed by G. H. Clevenger, Robert H. Bradford, H. W. Young.
Born 1889, Milwaukee, Wis. 1895–1907, Grammar and High School, Colton, Cal. 1908–13, Stanford Univ.; A. B. in Min. and Met. 1909, Nayal Mill. Co., Guanajuato, Mex. 1909, Guanajuato Amalgamated Mines Co. 1913, Eagle Shawmut Mines Co., Shawmut, Cal. 1913, Nevada Mill. Co., Tonopah, Nev. 1914, Colorado Min. Co., Aroroy, Masbate, P. I.
Present position: Asst. to A. D. Marriott, Jr., Dorr Cyanide Machinery Co.

Bernard Hugo van der Linden, San Francisco, Cal.
Proposed by Robert B. Moran, W. A. Williams, W. R. Hamilton.
Born 1880, Buitenzorg, Java. 1905, Univ. of Delft; M. E. 1906–07, Asst. in Petrography, Crystallography and Mineralogy, Univ. of Delft, Holland. 1907–13, Geol., Royal Dutch Oil Co., D. E. I.
Present position: 1913 to date, Chief Geol. and Field Mgr., Shell Company of California (California Oilfields, Ltd.).

Bruce R. Lindsay, Smithfield, Fayette Co., Pa.
Proposed by M. G. Moore, D. M. Stackhouse, Andrew B. Crichton.
Born 1883, Plymouth, Pa. 1890–1901, Plymouth Public Schools, Pa. Have first-grade mine-foreman certificate for bituminous mines of Pa. Mine-rescue certificate, Bureau of Mines. 1901–06, Employed on Engrg. Corps, Parrish Coal Co., Plymouth, Pa. 1906–08, Res. Engr., New River Collieries Co., Eccles, W. Va., in charge of mine work and shaft sinking. 1909, Dist. Engr., Lehigh Coal & Navigation Co., Lansford, Pa. 1910–15, Three years, Engr.; two years, Mine Supt., Cambria Steel Co., Johnstown, Pa.
Present position: Mine Supt., Republic Iron & Steel Co., Youngstown, O.

Hugh B. Lowden, Denver, Colo.
Proposed by S. A. Ionides, Richard A. Parker, J. W. Richards.
Born 1872, New York, N. Y. 1880–89, Public and private schools, and subsequently studied mathematics, chemistry, metallurgy, etc., privately. No degrees. Member, Colorado Scientific Society. 1890, Entered employ of Gates Iron Works, Chicago, Ill., manufacturers of rock and ore crushing, milling, and smelting equipment. 1905, Colorado Iron Works Co., Denver, Colo., metallurgy of the precious metals, copper and lead.
Present position: Secy., Colorado Iron Works Co.

George Caldwell McFadden, Witt, Ill.
Proposed by H. H. Stoek, E. A. Holbrook, H. E. Smith.
Born 1888, Lessmahage, Lanarkshire, Scotland. 1893–1901, Grade school, Danville, Ill. 1901–05, High school, Danville, Ill. 1905–08, Asst. Engr.; 1908–10, Div. Engr., Dering Coal Co., Westville, Ill. 1910–11, Div. Engr., Standard Collieries Co., Johnston City, Ill. 1911–12, Div. Engr., Burnwell Coal Co., Witt, Ill. 1912–14, Div. Engr., Peabody Coal Co., Witt, Ill.
Present position: 1914 to date, Div. Engr., Receivers, C. & E. I. coal properties.

Henry Douglas MacDonald, Mascot, Tenn.
Proposed by A. W. Koch, J. N. Houser, Robert Ammon.
Born 1887, Ludlow, Vt. 1907–08, Tufts College. 1908–13, Mass. Inst. of Tech., B. S. 1911, Min., Eustis Min. Co., Eustis, Que. 1913–14, Sampling, refining, diamond drill, Hollinger Gold Mines, Timmins, Ont.
Present position: 1914 to date, Asst. Met., American Zinc Co.

George Alexander Macready, Brighton, Trinidad, B. W. I.
Proposed by G. H. Clevenger, H. W. Young, D. M. Folsom.
Born 1885, Tacoma, Wash. 1906, Grad., Throop Polytechnic Institute, Pasadena, Cal. 1910, Grad., Leland Stanford Junior Univ.; A. B. in Geol. and Min. 1914, Grad., student in Geol., Univ. of Chicago. 1910–11, Resident Geol., North American Oil Consolidated, Taft, Cal. 1911, Instr., field and topographic Geol., Leland Stanford Junior Univ. 1911–14, Head Field Geol., Bermudez Co., and Caribbean Petroleum Co.; Venezuela.
Present position: 1914 to date, Resident Geol., Trinidad Lake Petroleum Co., Ltd.

W. Fred Nance, New Castle, Tex.
Proposed by George B. Rodgers, O. M. Bilharz, E. J. Rossbach.
Born 1884, Newnan, Ga. Grad., high school. Three years college course. Knowledge of engrg. acquired through experience and correspondence courses.
Present position: 1908 to date, Sec., Treas., Mine Supt. and Min. Engr., Belknap Coal Co.

Thomas Henry Oxnam, Jr., Fairview, Ariz.
Proposed by Philip Wiseman, Seeley W. Mudd, Frank A. Keith.
Born 1883, Lebanon, Ill. Preparatory school work, Los Angeles public schools, high school and Belmont Military Academy. 1905–06, Special work, Columbia Univ. 1897–98, Asst. Assayer, De Lamar Mines Co., De Lamar, Nev. 1901, 1903–05, Assayer and later Asst. Mine Supt., Palmarejo & Mexican Gold Fields, Mex. 1907–08, Chem., later Supt., for Receiver, Ariz. Smelting Co., Humboldt, Ariz. 1909, Underground miner, Copper Queen Mines, Bisbee, Ariz. 1909–10, General mill work, Goldfield Cons. and Combination Fraction, Goldfield. 1911, In charge of Min. Dept., Y. M. C. A. Technical School, Los Angeles, Cal. 1912–13, Met. Engr., Chino Copper Co., Hurley, N. M. 1914, Mine Supt., Gold Mountain Mines Co., Doble, Victorville, San Bernardino Co., Cal.
Present position: Met. Engr., Grand Reef Mill. Co.

Robert Dun Patterson, Lowe, W. Va.
Proposed by John J. Lincoln, L. E. Tierney, William D. Ord.
Born 1881, Dayton, O. 1904, Grad., Sheffield Scientific School, Yale University; Ph.B. Prepared for college at Taft School, Watertown, Conn. 1904, Came to Pocahontas coal field and assisted engr. of Weyanoke C. & C. Co., which place was just opening up. Later became Supt. 1912, Gen. Supt., Weyanoke C. & C. Co., White Star Min. Co., and Gulf Coal Co.
Present position: Genl. Mgr., Weyanoke White Star and S. J. Patterson Poca. Co.

John Painter Remick, Peking, China.
Proposed by Thomas T. Read, B. B. Thayer, J. W. Allen.
Born 1867, Ruabon, North Wales. To 1885, Various schools in England. 1885–87, Owen's College, Manchester. Student to J. & P. Higson, five years, out of which three years were spent in practical training at Hemsworth Collieries, Yorkshire and several collieries in Lancashire, Staffordshire, Cheshire, and North Wales. 1890–93, Asst., J. & P. Higson, Manchester. 1893–1907, Cons. Min. Engr. In charge of Flimby and Broughton Moor Collieries, Cumberland, Hockly Hall and Whaldey Collieries, Warwickshire, etc. Certificated Colliery Manager, 1st class, a member of the I. M. E. (British), Iron and Steel Institute, Institution Min. and Met. A. M. I. C. E.
Present position: General Agent & Engr., Pekin Syndicate, Ltd.

Charles Albert Schieren, New York, N. Y.
Proposed by Theodore L. Dwight, W. R. Ingalls, Richard H. Vail, W. L. Saunders, Frank W. Iredell, Leon P. Feustman.
Born 1869, Brooklyn, N. Y. 1879–1887, Polytechnic Institute of Brooklyn, N. Y. Traveled and founded agencies and branch houses in Europe, Cuba, Mexico, and all parts of U. S. A.
Present position: Pres., Charles A. Schieren Co.

Thomas McCourt Smither, Wonder, Nev.
Proposed by E. E. Carpenter, H. M. Alley, George J. Young.
Born 1892, Temple, Tex. 1900–05, Trinity School, New York, N. Y. 1905–06, Lowell High School, San Francisco, Cal. 1906–09, Univ. of Nevada. Preparatory School, Reno, Nev. 1909–13, Mackay School of Mines, Univ. of Nev., Reno, Nev.; B. S. in Min. Engrg. 1913–14, Montana-Tonopah Min. Co., Tonopah, Nev.
Present position: 1914 to date, Asst. Min. Engr., Nevada Wonder Min. Co.

William Leonard Stevens, Rancagua, Chile.
Proposed by Ross E. Douglass, Lester E. Grant, B. T. Colley.
Born 1878, Somerville, Mass. To 1896, Somerville, Mass., Public School. 1896–1900, Mass. Inst. of Tech.; S. B. 1900–01, Asst., Min. Dept., Mass. Inst. of Tech. 1901–05, Asst. Chem., Furnace Foreman, Genl. Foreman and Asst. Supt., Tenn. Copper Co. 1905–08, Supt., Tenn. Copper Co. 1908–10, Supt., Smelter, Copiapo Min. Co., Copiapo, Chile.
Present position: 1910 to date, Met. Accountant, Braden Copper Co.

John L. Templemen, Butte, Mont.
Proposed by W. D. Mangam, Horace V. Winchell, Fred T. Greene.
Born 1872, England. 1897, Iowa College, Grinnell, Ia.; A. B. 1899, Univ. of Virginia, Charlottesville, Va.; LL.B. 1915, Owner of and operating a lease of Nancy Hanks mine, Garnet, Mont. Owning and operating other properties in Montana of less consequence.
Present position: Attorney at law, Butte, Mont.

Arthur Thomas Ward, Sewell, Chile.
Proposed by B. T. Colley, R. K. Stockwell, Lester E. Grant.
Born 1890, Bellefonte, Pa. Until 1908, public and private schools, Bellefonte and Carlisle, Pa. Grad., Electrometallurgy, Lehigh Univ. Summers, 1908 to 1912, inc., sales dept., Aluminum Cooking Utensil Co., Phila. and New York Dist. 1913–14, Experimental Chem., U. S. Metals Refg. Co., Chrome, N. J.
Present position: 1914 to date, Experimental Engr., Braden Copper Co.

Francis Watson, Iquique, Chile.
Proposed by Huntington Adams, R. M. Atwater, Jr., C. R. Corning.
Born 1884, Lima, Peru. 1894–98, Barham House, a private school at St. Leonards-on-sea, England. 1898–1900, Harrow School, England. 1900–01, Junior clerk, Thomson Aiknan, Jr., nitrate of soda broker, London, E. C. 1901–03, Unauthorized clerk, Edward Lawrence & Co., London Stock Exchange. 1903–09, Clarke, Bennett & Co. of Iquique, Chile, employed in nitrate manufactories. 1909–10, Nitrate Agencies, Ltd., successors of Clarke, Bennett & Co., as manager of "Oficina" Santa Fè, Tocopilla.
Present position: Nitrate Agencies, Ltd., as Mgr., Nitrate Department, controlling 15 nitrate manufactories, and all exploration work.

Arthur Edward Wells, Berkeley, Cal.
Proposed by F. G. Cottrell, Robert H. Bradford, G. H. Clevenger.
Born 1884, Saxonville, Mass. 1890–1902, Public Schools, Framingham, Mass. 1902–06, Mass. Inst. of Tech.; B. S. in Min. Engrg. and Met. 1906–13, American Smelt. & Ref. Co. 1906, Chem., Globe & Arkansas Valley Smelters. 1907, Chem., Murray Smelter. 1908–09, Research Chem., Murray and Garfield Smelters. 1910–11, Asst. Met., Garfield Smelter. 1912–13, Met., Garfield Smelter. 1914, Met. in charge investigation, Selby Smelter Commission.
Present position: Met., Smelter Smoke Investigations, U. S. Bureau of Mines.

Frederick H. Willcox, Pittsburgh, Pa.
Proposed by Van H. Manning, A. H. Fay, W. A. Williams.
Born 1882, Smyrna, N. Y. 1902, Colgate Academy. 1906, Mass. Inst. of Tech.; B. S. 1906–07, Asst. Instr., Mass. Inst. of Tech. 1907–11, Duquesne Furnace, Carnegie Steel Co. 1911–13, National Furnaces, National Tube Co. 1913–14, Monessen Furnaces, Pittsburgh Steel Co. 1914, U. S. Bureau of Mines, Pittsburgh, Pa.
Present position: Met. Engr., U. S. Bureau of Mines.

William Henry Williams, New York, N. Y.
Proposed by Eli T. Conner, E. R. Pettebone, C. Dorrance, Jr.
Born 1874, Athens, O. Public school, Toledo, O. Business college, Beaver Falls, Pa. 1891–92, Engrg. Corps, P. & L. E. R. R. 1892–1901, Various positions, Penn. Lines W. of Pittsburgh. 1901–1904, Asst. Secy., Asst. to Genl. Mgr., B. & O. R. R. 1904, Supt. Frt. Trans., Frisco and C. & E. I. 1905–07, Traffic Mgr., Pittsburgh Chamber of Commerce. 1907, Asst. to Pres., Delaware & Hudson Co.
Present position: 1907 to date, Vice-Pres., Delaware & Hudson Co., and Hudson Coal Co.

Associate Members

Reuben Waldo Davis, Dolores, Webb Co., Tex.
Proposed by J. F. Kemp, Bradley Stoughton, M. C. Scheble, E. R. Jones.
Born 1877, Cuyahoga Falls, O. 1883–85, Public school, Cuyahoga Falls, O. 1885–87, Public school, Anson, Tex. 1887–92 Public school, Santo Tomas, Webb Co., Tex. 1892–95, International Correspondence Schools, Scranton, Pa. 1902–03, New York University, New York. 1903–04, Columbia University, New York. Attended several other schools for short periods between 1902 and 1905. 1894–95, Santo Tomas Coal Co. 1895–96, Rio Grande Coal Co. 1896–1902, 1905–07, Cannel Coal Co. 1907–09, Cia. Fundidora de Fierro y Acero, Cia. Carbonifera del Norte.
Present position: 1909 to date; Engr., Maintenance of Way, Rio Grande & Eagle Pass Ry.; Asst. Supt. and Min. Engr., Cannel Coal Co.

Paul Messer, Tokyo, Japan.
Proposed by W. L. Saunders, George A. Howells, L. D. Albin.
Born 1873, Chicago, Ill. 1878–87, Oakland public and high schools, Chicago, Ill. 1887–90, Chicago Manual Training School. 1890–94, Cornell Univ., M. E. Member, Sigma Xi Society, A. S. M. E., Japan Inst. of Min. Engrs. 1894–96, Asst. and Chief Engr., Fraser & Chalmers Co. 1896–97, Foreman, machine shop, Myrpacht, Randfontein Mine, So. Africa. Amalgamator, North Randfontein Mine. 1897–1903, Shop work, Fraser & Chalmers, erecting floor. Engrg. Salesman, Fraser & Chalmers and Allis-Chalmers Co., Chicago, Butte, Rossland, Spokane, New York.
Present position: 1903 to date, Mgr., Engrg. Dept., and Acting Agt., American Trading Co.

Philip Edward Thomas, Johnstown, Pa.
Proposed by M. G. Moore, D. M. Stackhouse, Andrew B. Crichton.
Born 1892, Johnstown, Pa. To 1911, Public schools, Johnstown, Pa. 1915, Princeton Univ.; B. S. in Geol.
Present position: Coal Sales Agent, Shade Creek Coal Co.

Felix Richard John Wanjura, Koskullskulls, Gellivare, Sweden.
Proposed by W. L. Saunders, George A. Howells, L. D. Albin.
Born 1870, Beuthen, Germany. 1880–89, Uppsala High School. 1889–92, Stockholm Technic High School. 1893–99, Verein Königs and Laurahütte-coal-mines, Laurahütte, Germany.
Present position: 1899 to date; Bergv. aht. W. C. Freja, Koskullskulls Iron Mine.

Junior Members

Thomas Edwin Nott, Pittsburgh, Pa.
Proposed by Roswell H. Johnson, H. B. Meller, S. L. Goodale.
Born 1892, Spartanburg, S. C. 1908, Grad., Gainesville, Ga., High School. 1908–10, Univ. of Georgia, Athens, Ga. 1910–11, Univ. of S. C., Columbia, S. C. 1911–13, Clinchfield Fuel Co., Spartanburg, S. C. 1913, C. C. & O. Ry., Erwin, Tenn. 1913, Clinchfield Coal Corp., Dante, Va.; 1913–14, Spartanburg, S. C.
Present position: Student, Univ. of Pittsburgh.

Changes of Address of Members

The following changes of address of members have been received at the Secretary's office during the period July 10 to Aug. 10, 1915. This list, together with the list published in *Bulletin* Nos. 100 to 104, April to August, 1915, and the foregoing list of new members, therefore, supplements the annual list of members corrected to Mar. 1, 1915, and brings it up to the date of August 10, 1915.

Allen, John A. .... Castano Nuevo, San Juan, Argentine.
Allen, Milton A., Mgr. .... Pearsite Co., Inc., Cannel City, Ky.
Allport, Robeson H. .... 724 W. Marshall St., San Antonio, Tex.
Ammon, Robert, Asst. Mill Supt. .... American Zinc Co. of Tenn., Mascot, Tenn.
Arms, Charles S. .... 5125 Wayne Ave., Philadelphia, Pa.
Bassett, Thomas E. .... Care Y. M. C. A., Bisbee, Ariz.
Batcheller, J. H., Supt., Ravenswood Mine, U. S. Smelting Co., Reeds, Jasper Co., Mo.
Behre, Henry A. .... Care Braden Copper Co., Rancagua, Chile.
Bell, J. M., Geol. and Min. Engr.; Cons. Engr., L. Ehrlich & Co., 310 Dominion Express Bldg., Montreal, Canada.
Bellinger, H. C., Care M. Guggenheim Sons, Room 3546, 120 Broadway, New York, N. Y.
Benham, Willard Miles .... Ray, Ariz.
Bernhard, Durward I. .... Managua, Nicaragua.
Berray, Niles S. .... 349 Carroll Park E., Long Beach, Cal.
Blackmar, Frank H. .... 706 Garfield Ave., Kansas City, Kan.
Blake, True Walter, Min. Engr. .... Davis Creek Coal Co., Novinger, Mo.
Boise, Charles Watson, Chief of Mines de la Forminière, Kasai, Tshikapa, Belgian Congo.
Boler, Charles A. .... 135 S. Second St., Philadelphia, Pa.
Braddock, Homer F. .... Mt. Pleasant, Pa.
Brandt, Maurice Sheldon, Chief Engr., White Eagle Mining & Milling Co., Boulder, Colo.
Brayer, N. G., Supt., Sharon Works, National Malleable Castings Co., Sharon, Pa.
Carpenter, Clark B. .... Ray, Ariz.
Carson, Joseph A. .... Care Allis-Chalmers Mfg. Co., Milwaukee, Wis.
Caruthers, A. Wayne .... Old Company's Club, Lansford, Pa.
Church, John L. .... Care Placer Hotel, Helena, Mont.
Crandall, Roderic .... 454 Forest Ave., Palo Alto, Cal.
Dake, Walter M., Jr., Met. .... 815 Gaylord St., Denver, Colo.
Defty, William E. .... 618 N. Third Ave., Phœnix, Ariz.
Dick-Cleland, A. F., Care Institution of Mining and Metallurgy, 1 Finsbury Circus, London, E. C., England.
Dixon, A. Faison, Field Mgr., Colon Development Co., Ltd., Camp Rio de Oro, Venezuela.
Eames, Luther B., Mill Supt., Hollinger Gold Mines, Ltd., Box 228, Timmins, Ont., Canada.
Enos, Herbert C. .... Carrizozo, N. M.
Evans, Cadwallader, Jr. .... R. F. D. 1, Sawtelle, Cal.
Eveland, Arthur J. .... Care Engineers' Club, 32 W. 40th St., New York, N. Y.
Ford, Harold Percy .... 2023 S. 20th St., St. Joseph, Mo.
Fulton, Chester A. .... 798 N. Broadway, Yonkers, N. Y.
Gard, Irving R., Consulting Engr., Columbia River Coal Dock Co., North Portland, Ore.

Gillespie, F. M. S....................................Abangarez, Costa Rica.
Goodland, Gillmore........Devonshire Club, St. James St., London, England.
Grant, Bertram..Kingman, Ariz.
Greenan, James Owen, Min. Engr.; Engr., Sarita Mining Co., Masonic, Cal., via Sweetwater, Nev.
Grether, Walter Scott, Min. Engr.; Asst. Min. Supt., Butte & Superior Copper Co., Ltd., Butte, Mont.
Griggs, Thomas W., Director, Colfax Consolidated Coal Co., Davenport, Iowa.
Hance, James H....................................202 Perry St., Elgin, Ill.
Harbach, Herbert M..............Care Lackawanna Club, Lackawanna, N. Y.
Harden, Edward B.........................Panama Canal, Washington, D. C.
Harder, Edmund Cecil, Geol......U. S. Geological Survey, Washington, D. C.
Hardy, J. Gordon, American Smelting & Refining Co., 120 Broadway, New York, N. Y.
Hess, Rush M..................Care Kenneth M. Hess, Cocoanut Grove, Fla.
Hogoboom, William Coryell, Drill Inspector, Southeast Missouri Lead Co., Potosi, Washington Co., Mo.
Holt, Theodore P..............Supt., Tintic Milling Co., Silver City, Utah.
Honnold, William L........5 London Wall Bldgs., London, E. C., England.
Hotchkin, Merritt W................Box 356, Haileybury, Ont., Canada.
Howard, Leslie E..........................259 Niagara St., Lockport, N. Y.
Howat, Andrew M., Min. Engr.; Supt. of Mine, Mogollon Mines Co., Mogollon, Socorro Co., N. M.
Hudson, Albert W., Met............Phelps, Dodge & Co., Douglas, Ariz.
Hutchinson, Robert M..............Care Lackawanna Club, Lackawanna, N. Y.
Inglis, J. F......... ....Care B. & H. Mine, Twin Bridges, Mont.
Jenks, Norman L........ ......Care Tracey Bros., Medellin, Colombia, S. A.
Jenks, Thomas H...........................Box 173, Albuquerque, N. M.
Johnston, C. W............................122 Florida Ave., Port Norfolk, Va.
Kennedy, Joseph E...........................120 Broadway, New York, N. Y.
Kidder, S. J, Genl. Mgr...............Mogollon Mines Co., Mogollon, N. M.
King, Paul S.................................701 W. Tenth St., Wilmington, Del.
Koehler, Carl F.............Youngstown Sheet & Tube Co., Youngstown, Ohio.
Kong, S. T.....................The "Riversite," Tungshan, Canton, China.
Liebig, John Orth............Balbach Smelt. & Refining Co., Newark, N. J.
Loerpabel, W. Harrison.......................116 Ellison St., Lead, S. D.
McKenna, Alexander G., Chem. and Met., Washington Steel & Ordnance Co., Washington, D. C.
McMeekin, Charles W...................1226 47th Ave., San Francisco, Cal.
Macfarlane, A. Kennedy, San Francisco Mines of Mexico, Ltd., Apartado 48, Parral, Chih., Mexico.
MacKay, Henry S..............MacKay Copper Process Co., Norwich, Conn.
Magnuson, Mark G........49 Chamber of Commerce, Minneapolis, Minn.
Marshall, James J.............................R. F. D. 3, Farmville, Va.
Marshall, S. M., Mgr., Turbine, Condenser and Pump Departments, Southwark Foundry & Machine Co., 5th & Washington Ave., Philadelphia, Pa.
Masters, Harris K..................392 E. Washington Ave., Bridgeport, Conn.
Maxon, Walter Landon...................60 Greenbush St., Cortland, N. Y.
Means, Ellison C.......................................Box 99, Rockville, Md.
Mitinsky, A. N., Min. Engr., Chief Engr. of Railroad supplies department, V. O. 2, Line N., 3, Petrograd, Russia.
Moore, Edward W.......................5343 Dorchester Ave., Chicago, Ill.
Moore, Elwood S.......................Dept. of Geology, State College, Pa.
Moore, L. D...............Care Darrow-Mann Co., 40 Central St., Boston, Mass.
Moule, J. W., Care Australian Institute of Mining Engineers, 57 Swanston St., Melbourne, Vic., Australia.
Munroe, Charles E., Prof. of Chemistry, George Washington Univ., Washington, D. C.
Newell, Arthur..Darby, Mont.
Newell, Frederick H....Head of Civil Engr. Dept., Univ. of Illinois, Urbana, Ill.
Nieding, Burton B..........................Perseverance, via Thane, Alaska.
Nighman, C. Edwin, Shift Boss, Leonard Mine, Anaconda Copper Min. Co., Butte, Mont.
Ormrod, John D, Pres.............Donaldson Iron Co., Emaus, Lehigh Co., Pa.
Palmer, William Frederick...Miami, Ariz.

Parker, Edward W., Director, Anthracite Bureau of Information,
1032 Miners Bank Bldg., Wilkes-Barre, Pa.
Perkins, H. C. ....1701 Connecticut Ave., Washington, D. C.
Peterson, Axel M. ....810 Peach St., Erie, Pa.
Phillips, Drury McNeill..................1220 Proctor St., Port Arthur, Tex.
Phillips, William B., Pres.......Colorado State School of Mines, Golden, Colo.
Pitman, S. M. ....Box 1118, Providence, R. I.
Pocock, Cecil W. ....Northern Ore Co., Edwards, N. Y.
Potter, Winfield S., Mfr., Pres..........Alloy Steel Forgings Co., Carnegie, Pa.
Prochazka, George A., Jr. ....Central Dye Stuff Chemical Co., Newark, N. J.
Read, Thomas T., Care Engels Copper Min. Co., Engels, via Keddie, Plumas Co., Cal.
Rhoads, Albert E. ....Frontier Brass Co., Niagara Falls, N. Y.
Riebel, Otto F. ....972 E. 27th St. N., Portland, Ore.
Robertson, Jasper T. ....9541 Winchester Ave., Chicago, Ill.
Rodgers, Joseph H. ....R. F. D. 10, Box 291, Los Angeles, Cal.
Rodgers, Selden S. ....Care Union Basin Mining Co., Golconda, Ariz.
Rogers, Benjamin C. ....Detroit Copper Mining Co., Morenci, Ariz.
Rork, Frank Curtis................Care Mine La Motte Co., Mine La Motte, Mo.
Rosewarne, D. D., Royal Colonial Institute, Northumberland Ave., London,
W. C., England.
Ryan, Edward C., Asst. Supt..............Federal Furnace Co., S. Chicago, Ill.
Schaber, Carl F., Div. Engr., Tennessee Coal, Iron & R. R. Co., Bessemer, Ala.
Schaefer, Frederick Rullman......Good Roads Machinery Co., Fort Wayne, Ind.
Scheble, M. C. ....Box 55, Sinton, Tex.
Schmalz, Charles H. ....130 Burlington Ave., Billings, Mont.
Semple, Robert Alexander.....................57 W. 53d St., New York, N. Y.
Senger, Richard Warren..........................Asarco, Durango, Mexico.
Sheafe, Harry J. ....Instructed to hold all mail.
Skogmark, John, Constr. Engr.............Mineral Paint Line Co., Chicago, Ill.
Smale, Frank L. ....Box 289, Victor, Colo.
Smith, Lloyd B. ....824 S. Detroit Ave., Tulsa, Okla.
Somers, Ransom E. ....804 E. Seneca St., Ithaca, N. Y.
Sonnemann, G. A. ....218 Riverside Ave., Spokane, Wash.
Sparks, James T. ....326 Erie St., El Paso, Tex.
Sproat, A. D., Cia. de Santa Gertrudis S. A., Pachuca, Hgo., Mexico, via Vera Cruz.
Starkey, Tom R. ....Penmaen, Hampton Wick, Middlesex, England.
Staver, W. H. ....Ophir, Colo.
Sweeny, Harry P., Genl. Mgr., Canadian Atlantic Coal Co., Ltd.,
Port Malcolm, N. S., Canada.
Thompson, Arthur P. ....Magdalena, N. M.
Thomson, Alexander T., Care Phelps, Dodge & Co., 99 John St., New York, N. Y.
Thomson, Edward.................Av. Vallarta 107, Guadalajara, Jal., Mexico.
Thorne, William E., Lenskoie Gold Mining Co., Ouspensky Mine, Bodaibo, Siberia.
Tibby, Benjamin F. ....4 Caithness Apts., Salt Lake City, Utah.
Tsai, Hsiang..........................Columbia University, New York, N. Y.
Wang, Chung Yu..............Tayeh Iron Mines, Tayeh, Hupeh Province, China.
Warner, Robert K. ....128 Oneida St., Syracuse, N. Y.
Warren, Frank M., Director, Asst. Sec., Treas., Engr., Pine Land Co.,
1224 First "National-Soo" Line Bldg., Minneapolis, Minn.
West, William C. ....Fort Myers, Fla.
Wicks, Frank R., Asst. Supt...............Chino Copper Co., Hurley, N. M.
Wilding, James, Met. and Min Engr., Mineral Separation American
Syndicate, 825 Merchants Exchange Bldg., San Francisco, Cal.
Wilson, Gordon.............................Apartado 7, Zacatecas, Zac., Mexico.
Witt, Herbert Nelson..............Care A. M. Ardery, Carson City, Nev.

Addresses of Members and Associates Wanted

Name Last address of Record, from which Mail has been Returned.
Boise, Charles Watson, Care Forminière, 8 Montagne du Parc, Brussels, Belgium.
Clerc, Camille...........................92 Rue Jouffrey, Paris, France.
Dowler, Harry P. ....Penn-Mary Coal Co., Heilwood, Pa.
Geiger, Arthur W. ....220 Mills Bldg., San Francisco, Cal.
Graves, McDowell......................949 Atlantic Ave., Long Beach, Cal.
Jones, Thomas J. ....Perm Govt., via Petrograd, Russia.

KIRKLAND, T. C., Cia. Met. y Refinadora del Pacifico S. A., Fundicion, Son., Mexico.
MCCARRICK, E.....628 N. Serrano Ave., Los Angeles, Cal.
MILLER, FRANK BARTON.....Blair, Esmeralda Co., Nev.
MORRIS, CHARLES E.....Pony, Madison Co., Mont.
MORSE, ROBERT G.....Mass Steel Casting Co., Everett, Mass.
OLIVER, ROLAND B.....8 Montagne du Parc, Brussels, Belgium.
PRICHARD, WILL A.....Room 711, 111 Broadway, New York, N. Y.
RHODES, W. B.....Golden, Colo.
RUBEL, MILTON L., Pilones Min. Co., Topia, Dgo., Mex., via Nogales
and Culiacan, E. de Sin.
SCHINDLER, DONALD F.....215 N. Murray St., Madison, Wis.
SEIBERT, PERCY A.....Hagerstown, Md.
SOMERS, RANSOM E.....804 E. Seneca St., Ithaca, N. Y.
STILLWAGON, SAM C.....329 W. 58th St., New York, N. Y.
WELSH, NORVAL J. E.....Organ, P. O., N. M.
WILCOX, RALPH.....Curanilahue, Chile.
WRIGHT, JESSE T.....Wallace, Idaho.

NECROLOGY

The deaths of the following members were reported to the Secretary's office during the period July 10 to Aug. 10, 1915:

Date of Election.	Name.	Date of Decease.
1886	* Foote, William Banks	June 25, 1915
1887	* Green, Charles B	Apr. 24, 1915
1895	* Holmes, Joseph A	July 12, 1915
1881	* Ormrod, George	June 21, 1915

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* Member

EXECUTIVE COMMITTEES OF LOCAL SECTIONS

New York

Meets first Wednesday after first Tuesday of each month.
DAVID H. BROWNE, *Chairman*, JOHN H. JANEWAY, *Vice-Chairman*.
F. E. PIERCE, *Secretary*, 35 Nassau St., New York, N. Y.
P. A. MOSMAN, *Treasurer*.
LEWIS W. FRANCIS.

Boston

Meets first Monday of each winter month.
HENRY L. SMYTH, *Chairman*. ALFRED C. LANE, *Vice-Chairman*.
HENRY A. WENTWORTH, *Secretary-Treasurer*, 60 India St., Boston, Mass.
ROBERT H. RICHARDS, ALBERT SAUVEUR.

Columbia

Holds four sessions during year. Annual meeting in September or October.
FRANK A. ROSS, *Chairman*. RUSH J. WHITE, *Vice-Chairman*.
LYNDON K. ARMSTRONG, *Secretary-Treasurer*, P. O. Drawer 2154, Spokane, Wash.
FREDERIC KEFFER, FRANCIS A. THOMSON.

Puget Sound

Meets second Saturday of each month.
I. F. LAUCKS, *Chairman*. J. F. MENZIES, *Vice-Chairman*.
GLENVILLE A. COLLINS, *Secretary-Treasurer*, Box 144, Seattle, Wash.
H. L. MANLEY.

Southern California

SEELEY W. MUDD, *Chairman*. C. COLCOCK JONES, *Vice-Chairman*.
FREDERICK J. H. MERRILL, *Secretary-Treasurer*, 631 Higgins Bldg., Los Angeles, Cal.
RALPH ARNOLD, A. B. CARPENTER.

Colorado

CHARLES A. CHASE, *Chairman*. S. A. IONIDES, *Vice-Chairman*.
C. LORIMER COLBURN, *Secretary-Treasurer*, 614 Ideal Bldg., Denver, Colo.
FRED H. BOSTWICK, W. G. SWART.

Montana

FRANK M. SMITH, *Chairman*. JAMES L. BRUCE, *Vice-Chairman*.
DARSIE C. BARD, *Secretary*, Montana State School of Mines, Butte, Mont.
FREDERICK LAIST, W. C. SIDERFIN.

San Francisco

Meets second Tuesday of each month.
G. HOWELL CLEVENGER, *Chairman*, C. W. MERRILL, *Vice-Chairman*.
JAMES C. RAY, *Secretary-Treasurer*, 1235 Webster St., Palo Alto, Cal.
F. W. BRADLEY, ANDREW C. LAWSON.

Pennsylvania Anthracite Section

R. V. NORRIS, *Chairman*.
CHARLES F. HUBER, *Vice-Chairman*, W. J. RICHARDS, *Vice-Chairman*,
EDWIN LUDLOW, *Vice-Chairman*, ARTHUR H. STORRS, *Vice-Chairman*.
CHARLES ENZIAN, *Secretary-Treasurer*, U. S. Bureau of Mines, Wilkes-Barre, Pa.
DOUGLAS BUNTING, FRANK A. HILL, ALBERT B. JESSUP.
RUFUS J. FOSTER, JOHN M. HUMPHREY, ROBERT A. QUIN.

St. Louis

ARTHUR THACHER, *Chairman*. R. A. BULL, *Vice-Chairman*.
WALTER E. McCOURT, *Secretary-Treasurer*, Washington Univ., St. Louis, Mo.
H. A. BUEHLER, R. R. S. PARSONS, HERBERT A. WHEELER.

Chicago

ROBERT W. HUNT, *Chairman*. J. A. EDE, *Vice-Chairman*.
HENRY W. NICHOLS, *Secretary-Treasurer*, Field Museum of Natural History, Chicago, Ill.
F. K. COPELAND, G. M. DAVIDSON.

Utah

R. C. GEMMELL, *Chairman*. C. W. WHITLEY, *Vice-Chairman*.
ERNEST GAYFORD, *Secretary-Treasurer*, 159 Pierpont Ave., Salt Lake City, Utah.
WALTER FITCH, WILLIAM WRAITH.

STANDING COMMITTEES

Executive

WILLIAM L. SAUNDERS, *Chairman*

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Membership

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Finance

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ALBERT R. LEDOUX,
CHARLES F. RAND.

Library

E. GYBBON SPILSBURY, *Chairman.*[4]

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BRADLEY STOUGHTON.

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BRADLEY STOUGHTON, *Chairman.*

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RENO H. SALES,
ALBERT SAUVEUR,
HENRY L. SMYTH,
A. A. STEVENSON
RALPH H. SWEETSER,
FELIX A. VOGEL,
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EDWIN C. HOLDEN,
FREDERICK W. SPERR,
GEORGE J. YOUNG,

WALTER R. CRANE, *Secretary*, Pennsylvania State College, State College, Pa.

LUTHER W. BAHNEY,
DARSIE C. BARD,
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SAMUEL W. BEYER,
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GEORGE S. RAYMER,
HEINRICH RIES,
HENRY L. SMYTH,
FRANCIS A. THOMSON,
F. W. TRAPHAGEN,
CLINTON M. YOUNG.

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[1] Until Feb., 1916. [2] Until Feb., 1917. [3] Until Feb., 1918. [4] Until Feb., 1919.

COMMITTEE ON INCREASE OF MEMBERSHIP

THOMAS T. READ, *Chairman.*
DAVID H. BROWNE, *First Vice-Chairman.*

Vice-Chairmen

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EDWARD H. BENJAMIN,
CHARLES A. BUCK,
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DUNCAN MACVICHIE,
VAN H. MANNING,
HENRY W. NICHOLS,
ERSKINE RAMSAY,
SUMNER S. SMITH,
WALTER G. SWART.

EDWARD L. DUFOURCQ, *Secretary,* C 22 Produce Exchange, New York, N. Y.

HUNTINGTON ADAMS,
F. L. ANTISELL,
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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Important Topping Plants of California

BY ARTHUR F. L. BELL,* SAN FRANCISCO, CAL.

(San Francisco Meeting, September, 1915)

PRIOR to 1908 the oil production in the State of California had been almost entirely a heavy fuel oil, with a high flash point, but changed within a short period to a large percentage of refining oil with a low flash, by reason of the coming in of the Santa Maria field, the Midway field, and the increased production of the Fullerton field.

This is clearly shown in the following tabulation of California's production:

Year	Barrels	Below 20° Bé, Per Cent.	Above 20° Bé, Per Cent.
1908	48,307,000	90	10
1910	77,698,000	80	20
1912	90,075,000	69	31
1914	103,600,000	47	53

The bringing in of these high percentages of light oil with their dangerous low flash and the increasing demand for the refined products necessitated the devising of some cheap and efficient means of quickly separating the lighter elements either in the field or at the main distributing points; also it became necessary to remove the excessive percentage of water which many of the light oils carried in emulsion.

UNION OIL CO.

Port Harford Plant

Probably the first important attempt to top light oil in California by other means than the ordinary stills was tried about 1908 by the Union Oil Co., at its plant at Port Harford, San Luis Obispo County, with Santa Maria oils. The process was the invention of Hubert G. Burroughs who afterwards obtained a patent for the process No. 998837, dated July

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*Chief Engineer, Associated Oil Company.

25, 1911. This consisted of a series of 12-in. pipes mounted in units of three pipes, one above the other, the crude oil being admitted to the upper pipe and filling the pipe about two-thirds of its diameter at which point it overflowed to the next 12-in. pipe below and in the same manner the oil again flowed to the third pipe below. Super-heated steam was the heating agent, which passed through endless coils in the lower half of the 12-in pipes. The oil was heated to about 200° F. in the upper pipe and reached a final heat of about 325° F. in the bottom pipe. The vapors generated passed off through the vapor line to the condenser.

The gravity of the Santa Maria crude, when the plant started, ran from 23 to 24 Bé, and averaged only from 2 to 3 per cent. water, but since then the average percentage of water has increased to 25 to 30 per cent.

This plant was operated for two or three years but the system developed several objectionable features. It was not economical, and by reason of the increase in water contents, the capacity was limited and was therefore abandoned.

Avila Plant

A new plant was designed and erected under the supervision of E. I. Dyer, Engineer in Chief of the Union Oil Co. This plant, of which A. Roy Heise is superintendent, is at Avila, near Port Harford, San Luis Obispo County, about 40 miles from the Santa Maria field, the oil being delivered to the plant from the field by two pipe lines; one 6 in. and the other 8 in. This plant, although erected several years ago, is still one of the best designed topping plants on the Coast. It is topping oils in some cases carrying as high as 20 per cent. water at the same cost for fuel and labor as some other plants topping oils carrying only 2 or 3 per cent. water.

The plant has a capacity of about 10,000 bbl. per day of clean oil or, as the oil is now running, 8,000 bbl. of net oil carrying about 20 per cent. water. The oil coming from the field carries as high as 30 per cent. of water, but a portion of it is directly treated in an electric apparatus, manufactured by the Petroleum Rectifying Co. of California, the oil thus treated being brought down to about 5 per cent., which when mixed with the rest of the oil coming from the field, gives an average of about 20 per cent. of water in the oil handled by the topping plant.

The heat used in topping is steam of 150 lb. pressure generated by a battery of five 300-h.p. Stirling boilers in a separate boiler house.

There are 12 cylindrical stills mounted in a steel-frame construction in a separate building, the stills being mounted in a double row of six each, one row above the other. Those in the upper set are known as the low-pressure stills, those in the lower as the high-pressure stills. Live steam is admitted to the lower row only. (Fig. 1.)

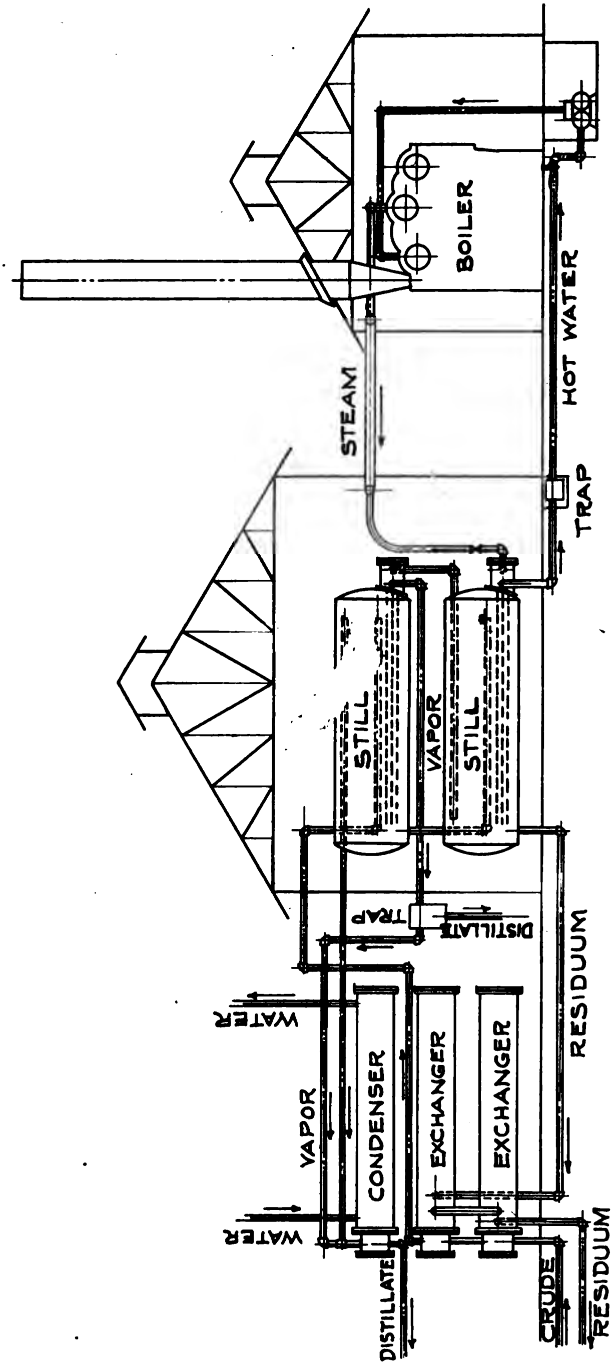


Fig. 1.—Union Oil Co. Plant, Avila, San Luis Obispo County, California.

Each still is about 6 ft. in diameter and 20 ft. long and has mounted on its front end, a steam chest, having three separate vertical compartments, the live steam being admitted to one compartment and passing to the next compartment through a series of 1-in. continuous pipes which travel from the initial side of the steam chest, the full length of the still and return to the next compartment of the steam chest, and in the same manner to the third compartment, so that there is no connection of live steam between any of the compartments of the steam chest except through the heating coils. The condensed water is trapped out of each compartment so that the minimum quantity of condensed water travels through the coils.

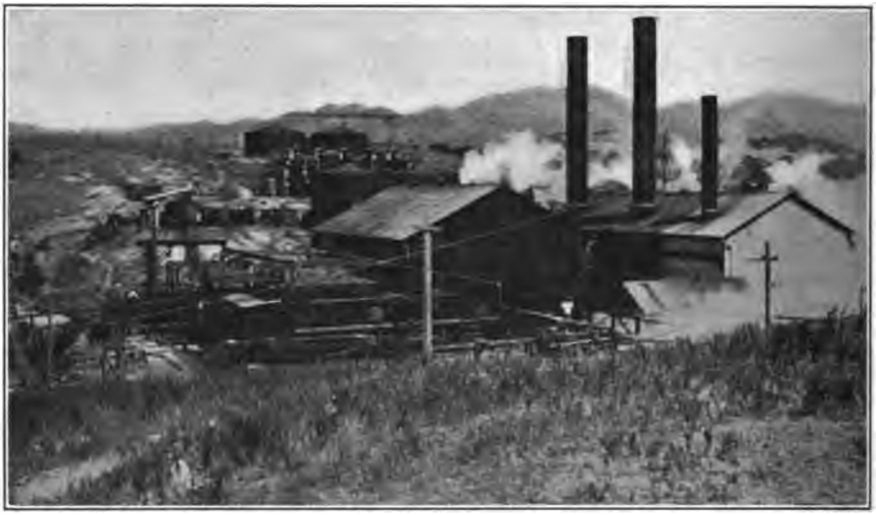


Fig. 2.—General View of the Avila Plant.

As the steam is condensed, it drains to traps and is returned to the boilers at 280° F. No live steam is admitted to the upper tier, or low-pressure stills, the heating agent being the evaporated vapors from the bottom still which rise and enter the steam chest of the upper still and pass through in the same manner as the live steam entered the lower still. The vapor pressure in both the lower still and inside the coils of the upper still ranges from 40 to 50 lb., the pressure being maintained at a point where the radiation of the latent heat through the coils to the liquid in the upper still is equalized by the heat vapors rising from the lower still. As about half of the vapor is steam, the temperature corresponds closely to that of saturated steam of the same pressure. No vapor leaves the coils of the upper still; nothing but the condensed liquids are trapped off, which go to the same condenser as the vapors from the low-pressure still. By this means every degree of latent heat in both the oil and water vapors rising from the high-pressure still is absorbed by the oil and water evaporated from the low-pressure still, thereby effecting a saving of about 50 per cent. of the fuel necessary to top the oil in ordinary stills. The vapors from each low-pressure still flow into the condenser belonging to that battery, which is mounted on the top of a steel construction outside of the steam-still house.

There are six separate condensers, each connecting to a battery of a high- and a low-pressure still. The general design of the condensers is rectangular with a vapor chest at the rear end of the same general design as that of the stills, except that the dividing partition in the chest is horizontal. The vapors are admitted to the upper portion of the chest and pass through a series of 2-in. return pipes, surrounded by the condensing water, and return to the lower portion of the chest, from there draining to the proper storage tanks.

Mounted below each condenser are two heat exchangers set one above the other. These heat exchangers are similar in construction to the condenser except that they have 1-in. return pipes, and have two vertical dividing partitions in each chest; also heated residuum surrounds the pipes instead of water. The heated residuum from the high-pressure still passes into one end of the body of the upper exchanger and travels backward and forward four times, guided by interior partitions. It then passes into the end of the body of the lower exchanger, again traveling backward and forward four times before passing out as finished residuum to storage. In all condensers and exchangers, the liquids counterflow.

The crude oil in taking up the heat in the exchangers comes into one side of the chest of the lower exchanger, passes backward and forward twice through a series of 1-in. return pipes and then is discharged from the opposite side of the chest to the upper exchanger. It passes through two series of return pipes in the upper exchanger, leaving the chest on the opposite side from where it entered. It flows to the low-pressure still, and after giving off a proportion of its vapors of oil and water, is pumped to the high-pressure still where all of the remaining water and distillates are taken off. The residuum oil passes to the exchangers.

When I examined these stills, the pressure of the vapors rising from the low-pressure stills was 3.6 lb. with a temperature of 183°F. In the high-pressure stills, and inside the coils of the low-pressure stills, the vapor pressure was 48.3 lb. with a temperature of 279° F.

This is a most economical system, since the only steam admitted to the plant is through the high-pressure stills; all of the work done in the low-pressure stills is therefore an absolute saving.

The plant is running on what is known as double effect, but at times has been run on triple effect, that is, live steam was introduced into the first still only and the heated vapor from the first still under pressure was admitted to the coils of the second still, and the heated vapor from the second still, necessarily under a less pressure, was admitted to the coils of the third still, the crude oil flowing first to the low-pressure, then to the intermediate, and finally to the high-pressure still.

Distillation by Double Effect

	Water, Per Cent.	Distillate, Per Cent.
Evaporation from low-pressure still	56.95	34.45
Evaporation from high-pressure still	42.05	64.15
Evaporation, Interchanger and Unaccounted.	1.00	1.40
	100.00	100.00

This method, as would be expected, showed a greater saving than the double effect, but reduced the quantity of crude that could be put through in a given time:

Distillation by Triple Effect

	Water, Per Cent.	Distillate, Per Cent.
Evaporation from low-pressure still	55.80	37.42
Evaporation from intermediate still	29.75	31.36
Evaporation from high-pressure still	13.42	29.74
Evaporation from interchanger and unaccounted.	1.03	1.48
	100.00	100.00

From the above figures, it will be seen that with the double-effect arrangement about 50 per cent. of the fuel is saved, and for triple effect about 75 per cent., over a single system of evaporation.

For a given period, the average temperature of the crude entering the heat exchangers was 61.67° F.; leaving the heat exchangers before entering the low-pressure stills, the temperature was 141.45° F., showing a saving by rise in temperature due to exchangers of 79.78° F.

The economy of the plant is shown by the following:

Duty Records per Barrel of Dry Fuel Oil Used for the Last Three Months of 1914

Barrels of crude oil treated	63.35
Barrels of water actually removed from crude	10.67
Barrels of residuum produced	42.34
Barrels of distillate produced	8.54
Barrels of residuum pumped from plant	43.80
Barrels of distillate pumped from plant	9.52
Barrels of circulating water pumped	156.46
Barrels of miscellaneous oil pumped	9.08
Kilowatt hours—lighting, power, etc.	2.53

Thermal Duty of 1 *bbl. of Dry Fuel Oil*

	B.t.u.
Evaporating water from crude	4,135,984
Evaporating distillate from crude	425,128
Kilowatt hours—lighting and power	35,940
Total useful duty	4,597,052

The average percentage of water in the crude was 18.71 per cent.

A record test run on "Double Effect" showed the following results:

	Per Cent.
Fuel consumed per net barrel of:	
Crude treated	1.75
Residuum manufactured	2.51
Distillate manufactured	11.70
Water evaporated	11.58
Both water and dist. evaporated	5.82
Water in the crude oil	15.38

This system is ideal for Santa Maria oils and entirely eliminates the coking which takes place when direct heating is used under ordinary stills or retorts. The water carried by the oil is as salty as sea water and when evaporated throws down salt crystals in the stills. In the case of fire-heated stills, this salt mingles with the coke and forms a hard incrustation, which necessitates closing down frequently and chipping out the coke; whereas with this form of still, when salt crystals are formed, the operators employ the unique method of shutting down the stills about once a week, emptying the stills of oil and filling with hot water, thus dissolving the salt and passing it off by pumping a continuous stream of heated water through the stills. Two patents on this system have been applied for by Mr. Dyer and are now pending in the patent office.

Brea Refinery

The Union Oil Co. has two other topping plants located in the Fullerton field. The most important is the Brea refinery. The first portion of this plant was completed in August, 1911, to handle the light oils from the Birch well, which came in with a very gassy oil flowing about 2,400 bbl. per day. It consisted of eight old 40-h.p. drilling boilers, with half the tubes removed; these boilers were set in two rows of four facing each other with an alley between, the oil being run through each battery of four boilers continuously. The plant when first started treated successfully about 100,000 bbl. per month of clean dry oil carrying not more than 0.5 per cent. of water, removing 22 per cent. of 53° tops. However, it has since developed that this plant is not adapted to handling wet oils and will

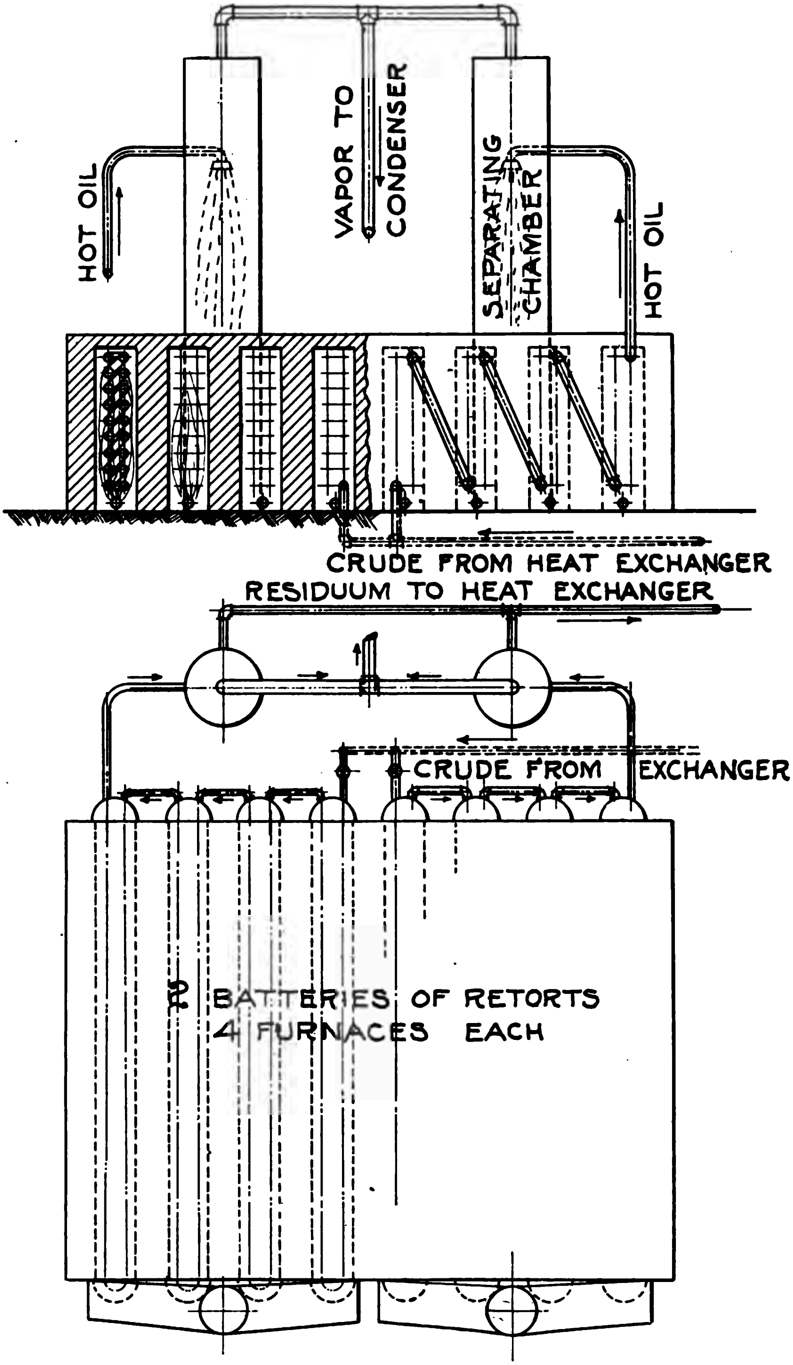


FIG. 3.—TOPPING PLANT AT BREA REFINERY, UNION OIL CO.

not work economically with oils carrying more than 2 per cent. water. It is therefore kept running on the cleaner oils.

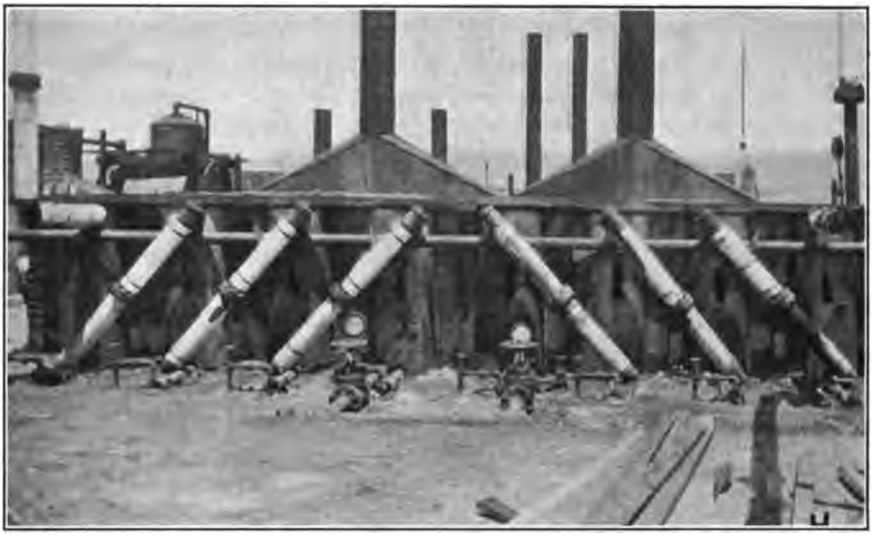


FIG. 4.—FRONT VIEW OF BREA TOPPING PLANT.

The next step in increasing the capacity of the plant was the building of an additional topping plant in March, 1913. This consisted of two batteries of retorts, each battery being divided into two separate divisions, each division having four furnaces and each furnace carrying a double row,

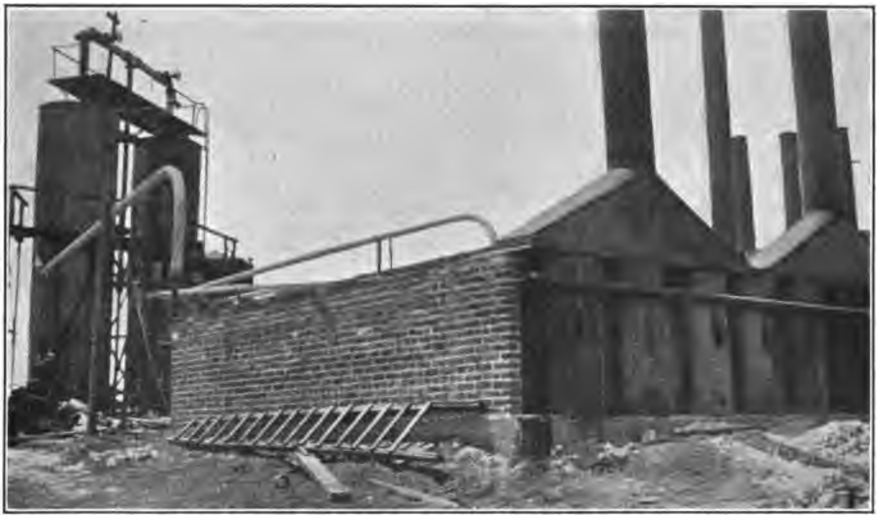


FIG. 5.—REAR VIEW, SEPARATORS IN BACKGROUND.

9 pipes high, of 3-in. return pipes, 20 ft. long. Each division of the battery is independently fed with crude oil. The oil admitted to the bottom pipe of the furnace travels up through the nine coils, coming out the top and returning to the bottom pipe of the next furnace, again leaving the upper pipe and so on through the four furnaces. The heated oil from each side of the battery flows by independent pipes to one of the separators. This makes each 8-furnace battery and one separator a single unit.

The vapors are then led from the separating chambers to the condensers and on to the storage tanks.

The residuum is taken from the bottom of the separators, passed through the heat exchangers and on to the residuum storage.

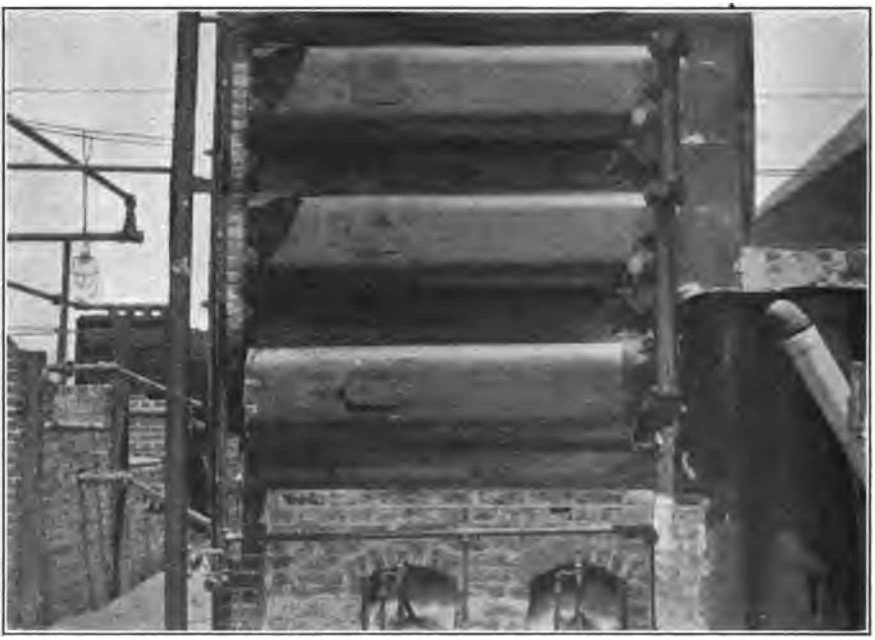


Fig. 6.—Rearranged Furnace for Oil Burning.

The coils in the furnace are heated by gas fed through perforated pipes running the full length of the furnace. It was found that this form of furnace is not well adapted to oil firing. A change was, therefore, made in one of the batteries, the coils being mounted horizontally in a brick furnace, allowing plenty of combustion room below. This furnace is working very satisfactorily and it is likely that the other half of the battery will have the coils changed from vertical to horizontal position so as to be adapted to oil firing. (Fig. 6.)

The total capacity of the two batteries of retorts with gas fires was 60,000 bbl. per month of 21 to 22° Bé. oil, carrying 6 per cent. B. S. & M., from which was topped about 10 per cent. of 51 gravity distillate, giving a residuum of 18° Bé. and from 160 to 180 flash. With the change in construction and the use of oil firing, it is believed the capacity will be doubled.

The hot oil at the outlet end of the retort is choked down to a pressure of 40 lb., but flows into the separator at practically atmospheric pressure. This choking of the oil gives a back pressure at the inlet end, due to friction in the retort, of 80 lb.

The crude oil leaves the heat exchanger and enters the first coil of the retort at 180° F.; it leaves the coils and enters the separators at about 450° F.

This refinery not only tops oil but turns out finished gasoline, but as the purpose of this paper is to describe topping plants only, no description will be given of the other details of the plant.

Naranjal Topping Plant

The Naranjal reduction or topping plant of the Union Oil Co., is in section 7, T. 3 S., R. 9 W., in the Fullerton field. This is the smallest of the three Union topping plants, having a maximum capacity of 2,200 bbl. per day. The oils treated are 16° to 18° Bé. and tops recovered 56° Bé. The plant has been built about 4 years and is known as a Brown-Pickering topping plant, it having been designed by A. H. Brown, the Superintendent. (Figs. 7, 8, 9.)

The plant consists of two independent units, each unit having a rectangular still about 4 by 4 by 20 ft. with cover bolted on and bottom of still fitted with cross baffles. The still is set, with a fall of about 6 in., over a brick furnace in which are placed 13 3-in. retort pipes, having return bends so as to make a continuous coil. The hot gases from the still furnace pass through an underground flue to a 40-h.p. drilling boiler, bricked up in the ordinary manner, these gases passing first under the boiler then through the tubes and out of the stack. In addition, the ordinary field type of a heat exchanger is used, made of a small pipe inside of a large one; for cooling the vapors, a series of wooden condenser boxes with pipe coils inside are provided.

In operating the plant, the crude oil is pumped through the inner pipe of the heat exchanger, the hot residuum flowing between the two pipes. The partially heated crude oil passes through the boiler, heated by waste gases, and from there through the coil retort in the furnace under the rectangular still, then into the high end of the still, where it is released and allowed to flow back and forth guided by the baffles until it reaches the lower end of the still, from which it flows by pipe through the heat exchanger to the residuum storage. As the oil flows through the still, which is heated by fire on its under side, the vapors pass off to a separating chamber, where the heavier vapors condense and are drawn off, and the lighter vapors pass to the condensers.

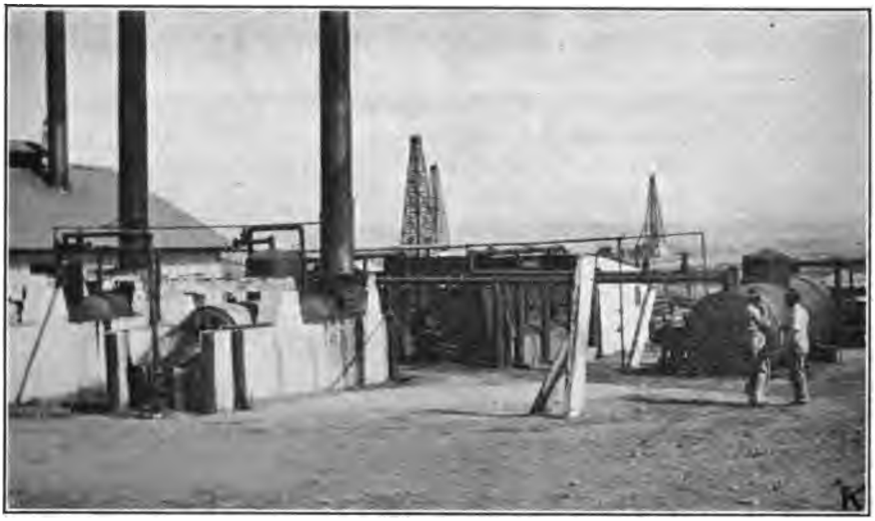


FIG. 7.—SIDE VIEW OF THE NARANJAL PLANT, SHOWING BOILERS AND STILLS.

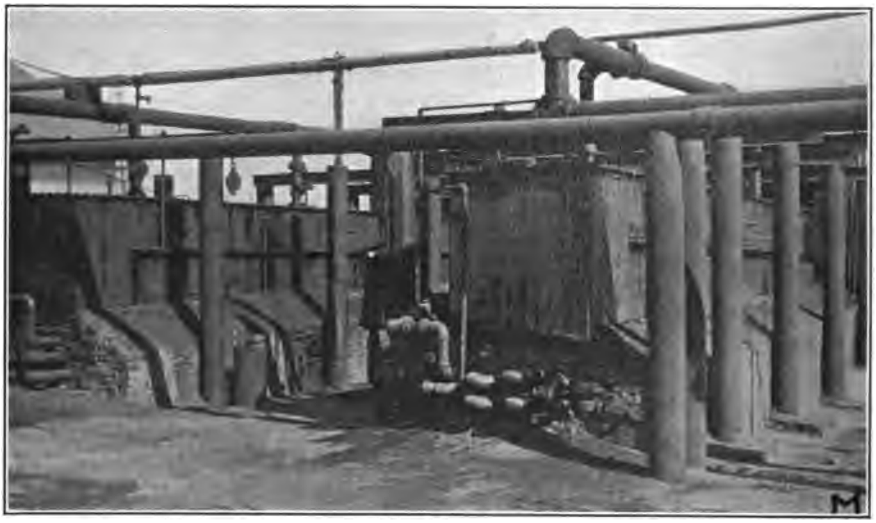


FIG. 8.—STILLS.

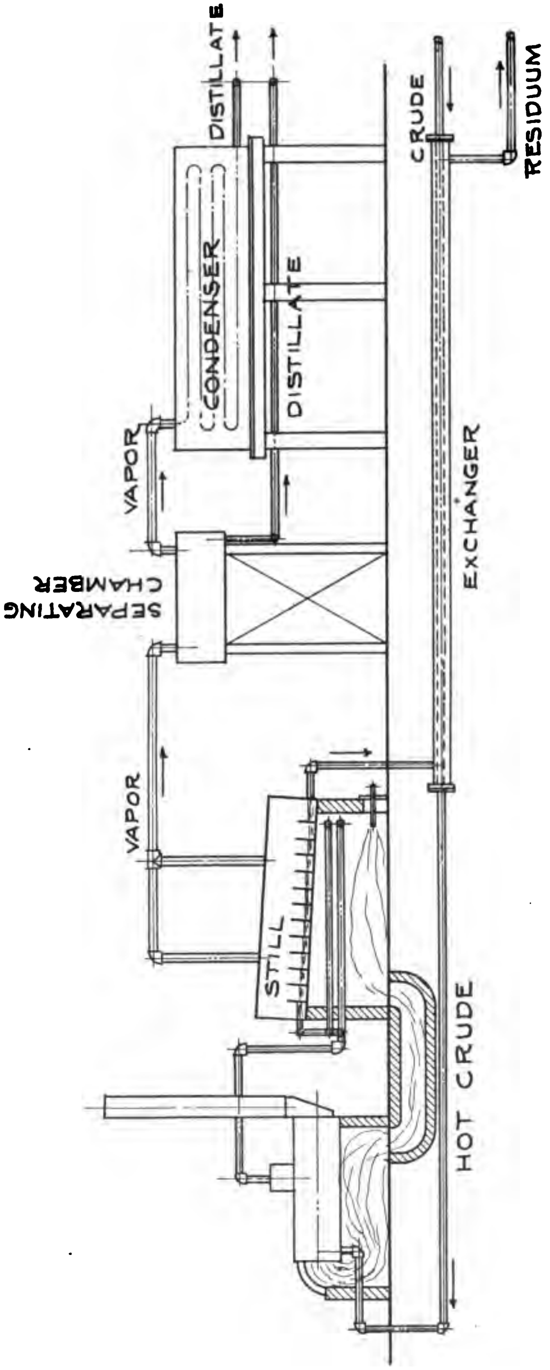


FIG. 9.—PLANT OF THE UNION OIL CO., NARANJAL, FULLERTON FIELD. BROWN-PICKERING DESIGN.

Each unit is a duplicate of the other, but is run entirely separate up to where the vapors enter the separating chamber; from there on the mixed vapors pass to the condensers. This system carries the notice "Patent applied for."

A number of these Brown-Pickering plants have been constructed in the oil fields and have given fair results with clean oil. They are not well adapted for wet oil, for as soon as they are worked to capacity with wet oil foam appears, which discolors the distillate, and the construction of the still does not admit of putting any pressure on it to hold down the foam.

Standard Oil Company

On the Murphy, Coyote Hills, property, Fullerton field, since bought by the Standard Oil Co., is one of the Brown-Pickering topping plants, consisting of four rectangular stills about 4 by 4 by 10 ft., each set with a gradual fall so that the oil admitted into the upper end of the first still flows down to its lower end and out into the upper end of the next still, which is set beside it and so on until the oil had flowed by gravity through the four stills.

At the rear end of the four stills is erected one 40-h.p. boiler so constructed that the hot gases from the four still furnaces pass under the boiler and through the tubes and out of the boiler stack. The vapors from the first two stills were run together and kept separate from the vapors coming off the third and fourth stills which were also run together. The vapors were then each run through separate pipe heat exchangers, the vapors in the inside pipe and the crude between the inner and outer pipes.

The continuation of the inside vapor pipe was then returned toward the plant and surrounded by an outer pipe this time filled with water, this forming the condenser for the vapors. The condensed vapors were then run to storage tanks.

In operating, the oil was first pumped through the outside of the vapor heat exchanger, then through a residuum heat exchanger, then to the boiler and from there to the upper end of the first still, then out of the lower end of the last still to residuum storage.

This plant had a capacity of about 1,200 bbl. per day of clean oil and 600 to 650 bbl. of wet oil carrying 25 per cent. water. If this latter capacity was exceeded, foaming took place.

With the crude oil entering the vapor heat exchanger at about 60° F., it entered the boiler heater at 80°, and left the boiler to enter the first still at 90° F.

The temperature of the oil on leaving the four stills ran 175 ° F., 200° F., 220° F., and 240° F., respectively. The fuel consumed was gas.

After the Standard Oil Co. purchased the property, the plant was used as a dehydrator, but was found so expensive that a 6-treator electric plant of the Petroleum Rectifying Co. was installed, which cleans the oil more satisfactorily and at a much less cost.

RICE RANCH OIL COMPANY

Another Brown-Pickering plant of the same type, but of a single unit, has been erected by the Rice Ranch Oil Co. in the Santa Maria field, Santa Barbara County. This plant is used mostly as a dehydrating plant but has also been used for topping. It was erected about 1911 and has worked very satisfactorily. Its capacity is claimed to be about 650 bbl. of crude carrying 3 per cent. water, but it also handles, at a reduced capacity, oil carrying as high as 30 per cent. water. The course of the oil through the plant is the same as in the other Brown-Pickering plants.

AMERICAN OIL FIELDS COMPANY OF CALIFORNIA

Another important topping plant is that erected by the American Oilfields Co., of California, about 1910, on section 36–31–22, in the Midway field. This plant has an economical capacity of from 18,000 to 20,000 bbl. of crude per day. It is arranged with three batteries, each battery consisting of eight independent retorts, making 24 retorts used on crude oil. Each battery delivers its heated oil to a single separator, where the

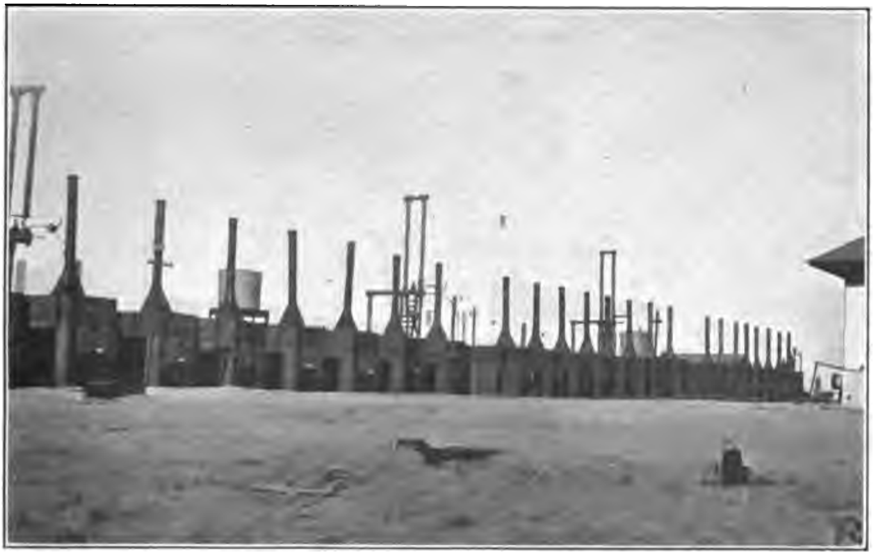


FIG. 10.—PLANT OF AMERICAN OIL FIELDS CO., SHOWING FRONT OF RETORTS.

vapors are separated, the vapors being led to a common header which distributes them to a series of independent condensers. (Figs. 10, 11, 12.)

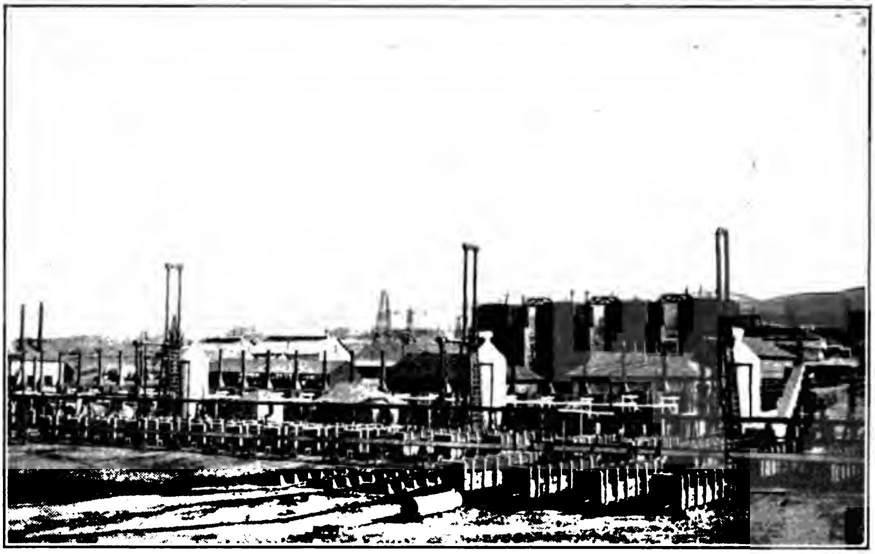


Fig. 11.—Rear View of American Oil Fields Plant, Water Cooling Towers in Background.

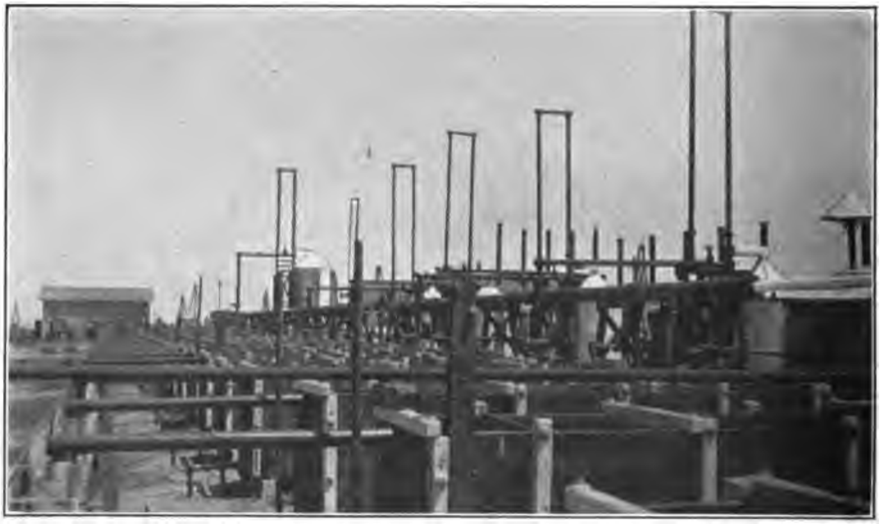


Fig. 12.—Condensers of American Oil Fields Plant.

This plant is well constructed and economically run and the company is to be complimented for its neat appearance.

All the condensing water is saved and run back through four independent cooling towers, which have a remarkable efficiency, since the circulating water after passing through these towers is cooled far below the temperature of the atmosphere. When I took observations, the water entered the towers at 140° F. and was taken out of the bottom at 65° F. with the surrounding temperature 99° F.

The plant is not only well adapted to topping clean crude, but is often called on to handle oils carrying as high as 25 and 30 per cent. water.

The fuel used at the refinery is natural gas.

PLANT OF SANTA FE RAILROAD CO., FELLOWS

The first really successful Trumbull plant, and the one that first attracted attention was erected at Fellows, Kern County, for the Santa Fe R. R. Co., to top its light oils so as to make safe fuel for locomotives. This plant consisted of two units of six 12½ in. by 20 ft. retorts, both units discharging into the one vapor separator.

The records of this plant for January, 1912, showed for 31 days' operation, with one of the units off for 5 days:

	Barrels	
Crude delivered to the plant	120,760.67	21° Bé.
Tops taken off	14,772	12.25 per cent.
Gasoline recovered	36.60	0.03 per cent.
Fuel consumed	1,575	1.34 per cent.

The Santa Fe R. R. Co. has since contracted its light oils to the General Petroleum Co. and has, therefore, closed down this plant.

PACIFIC CRUDE OIL CO., MIDWAY FIELD

The Pacific Crude Oil Co. built a small Trumbull plant on its property near Fellows for the purpose of topping oil in the field. Subsequently the company contracted its oil, so the plant was never started up. A good example of a small Trumbull topping plant is illustrated in Figs. 13 and 14.

GENERAL PETROLEUM CO., VERNON

The General Petroleum Co., in 1912, completed an 8-in. line from the Midway oil fields, Kern County, to San Pedro, Los Angeles County, and as the oils transported were too light and valuable for fuel purposes, it was necessary that a topping plant be erected at or near the terminal. The company adopted the Trumbull system of topping and erected a plant of 15,000 bbl. daily capacity (Figs. 15 to 19A) at the town of Vernon, in the outskirts of Los Angeles. All incoming oil is topped here and the residuum pumped to San Pedro. However, while this plant has a rated capacity of 15,000 bbl. daily, it has since been determined that the most economical capacity of the plant is about 12,000 bbl.

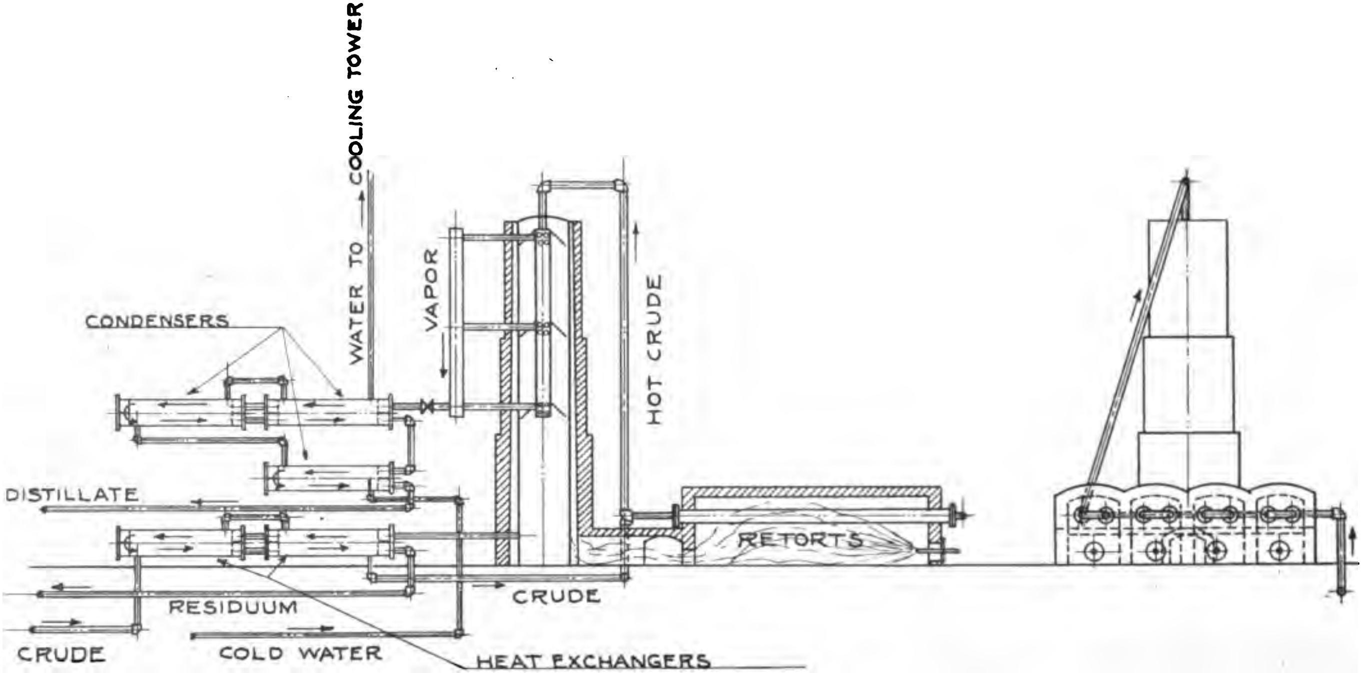


FIG. 13.—SMALL TRUMBULL TOPPING PLANT, PACIFIC CRUDE OIL CO., MIDWAY FIELD, KERN COUNTY, CALIFORNIA.

The General Petroleum Co., in adopting the Trumbull system, purchased a half interest in the Trumbull patents which to date are as follows: No. 996736, patented July 5, 1911; No. 1002474, patented Sept. 5, 1911; No. 1070301, patented Aug. 12, 1913; also patents secured in 72 foreign countries.

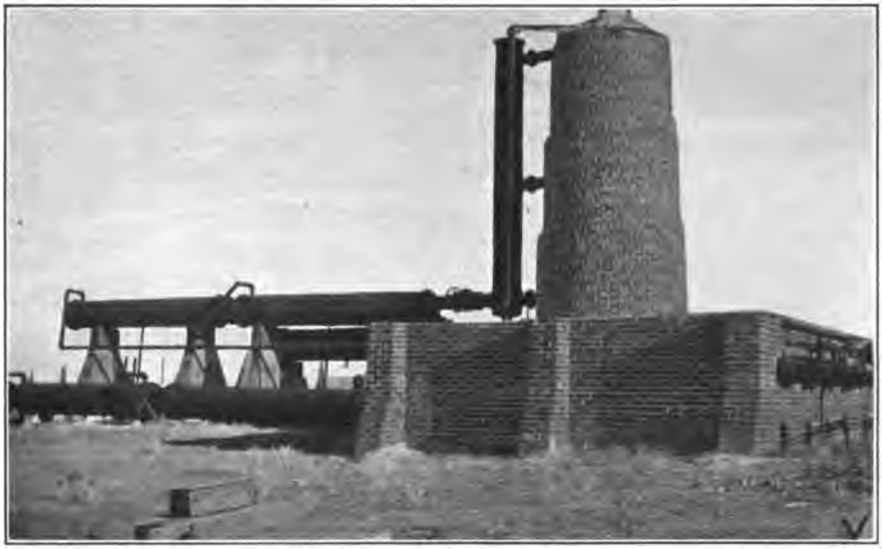


FIG. 14.—TRUMBULL TOPPING PLANT OF PACIFIC CRUDE OIL CO.

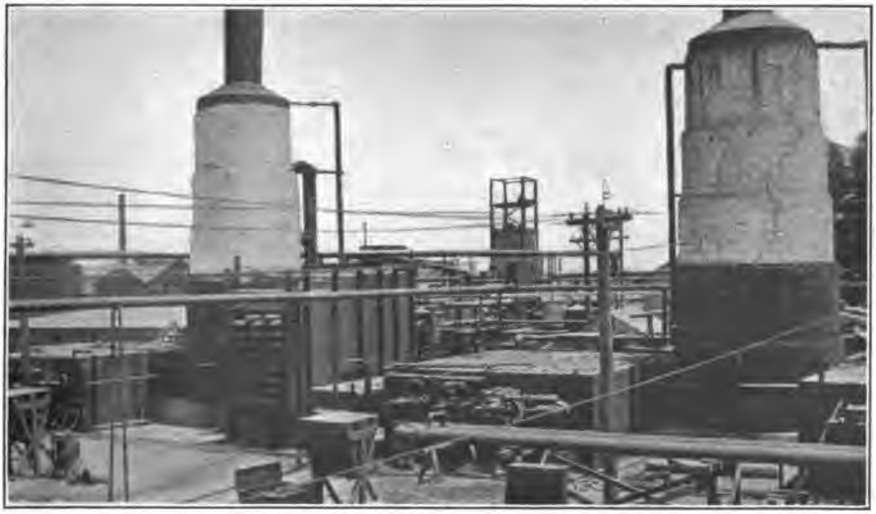


FIG. 15.—FRONT VIEW OF TRUMBULL PLANT OF GENERAL PETROLEUM CO.

The special features claimed for the Trumbull process are:

The saving of heat in the evaporating chamber by means of conical spreaders in the evaporators, all oil being forced to run down the inside surface of the shell, which is heated on the outside by the escaping flue gases.

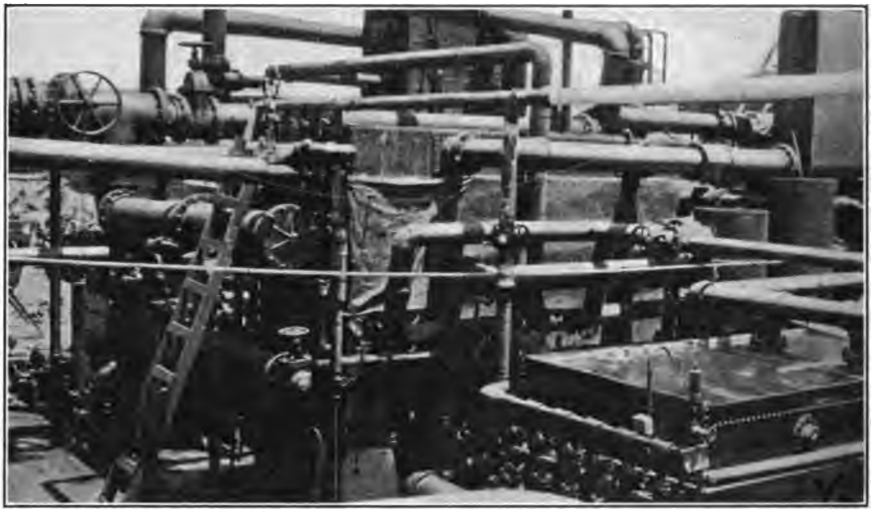


FIG. 16.—SEPARATORS.

The condensing of the first-run vapors and the re-evaporation of these distillates by passing them through a specially designed separator, which derives its heat from the outflowing first-run vapors.

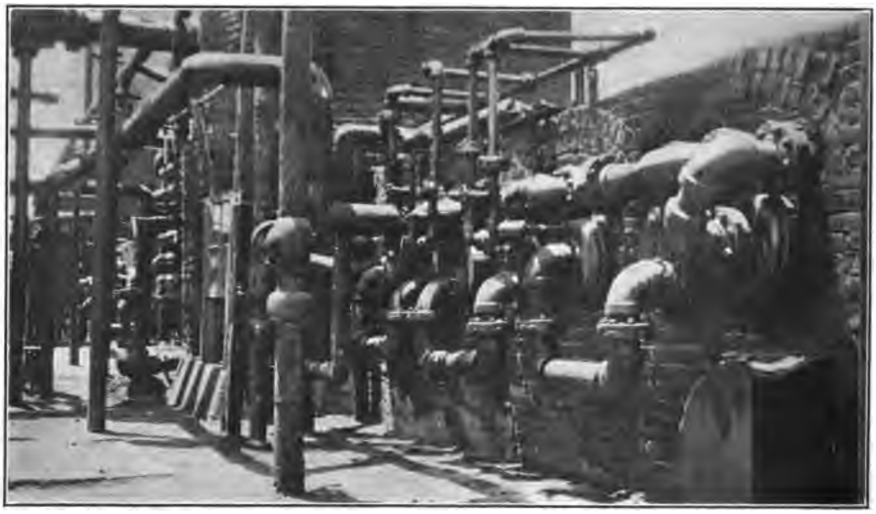


FIG. 17.—RETORT.

And the recovery of the re-evaporated distillates from separate compartments of the separator thereby furnishing different gravities of finished products.

The Vernon plant was completed in 1912, at a cost of about $80,000 and has been in continuous successful operation since that time. The plant consists of two independent batteries, each battery having two retorts discharging into a single evaporating chamber.* Each battery has two separator boxes, where the condensed distillates are re-run into finished products; also three heat exchangers about 4 ft. in diameter with 18-ft. tubes; two first-run vapor condensers of approximately the same size; and four re-run vapor condensers 30 in. in diameter with 18-ft. tubes.

In addition to the condensers, there are water-jacketed coolers consisting of one or two small pipes laid within a larger pipe for the purpose of further cooling the distillates before going to the storage tank.

In this plant, tubular condensers and water-jacketed pipe coolers have been used in every case instead of the ordinary condensers generally used about a refinery.

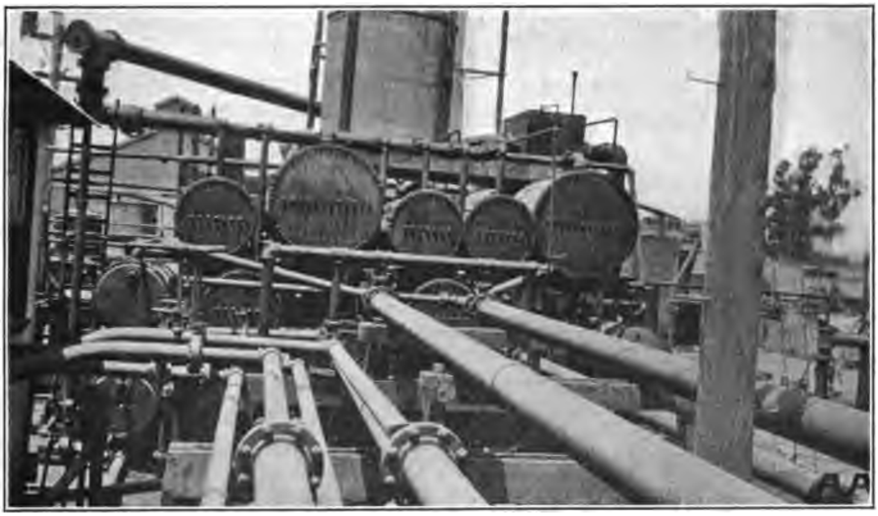


Fig. 18.—Vapor Condensers and Pipe Jacketed Coolers.

Since the plant was built, it has been found advisable to make several changes and additions; one important change being in the retorts. The original retort consisted of six 12½-in. pipes about 20 ft. long connected together at their ends with 4-in. pipe returns. Vent pipes were inserted at one end of each retort to lead off gases which, if trapped in the large pipes, would cause unequal heating and sagging of the retorts. It was found that there was not heating surface enough, the escaping gases in the stack running as high as 1,200° F. Additional 4-in. retort pipes were, therefore, placed above the 12½-in. pipes, thereby increasing the hetaing surface about 50 per cent., the result being that the temperature of the escaping gases was reduced to about 650° F. (Fig. 17.)

---

* This chamber is called a "separator" on all the plants described except in connection with the Trumbull plants where it is called an evaporating chamber.

The company has lately torn down one of the retorts and rebuilt it entirely of 4-in. pipe. This change has resulted in a marked improvement.

Another important feature added to the original construction is a large rectangular dephlegmater and re-evaporator erected at each unit through which all the first-run vapors pass. This dephlegmater, it is claimed, has improved the quality of the products by removing the heavier distillates which originally discolored them.

The oils handled average 20° to 21° Bé.; the residuum manufactured, 16° to 17° Bé.; the tops manufactured, No. 1 of both 55° and 58° Bé., No. 2 of 45° Bé. and still bottoms of 35° to 37° Bé.

Special cuts of various gravities are also turned out to order.

Temperatures Maintained at Plant and Percentages of Oil and Water

Crude run to heat exchangers	80° F.
Heat exchangers to retorts	264° F.
Out of retorts	445° F.
Residuum to storage	190° F.
Water in oil, per cent	0.5
Fuel consumed, per cent	1.6

The last item does not include oil consumed in pumping about the refinery, this being done from the main pipe-line pumping station, and by auxiliary electric pumps. The above, however, includes pumping of circulating water through the cooling towers.

The course of the oil through a single battery of the Trumbull plant at Vernon is as follows:

The crude, previously heated by passing through heat exchangers *J* and *H* (Fig. 19), is forced through the retort pipes. From there, it passes upward by a single pipe to the top of the evaporator *A*, and is sprayed over a conical spreader which directs the fluid against the inner surface of the shell. In its downward course, this heated crude comes in contact with additional conical spreaders, continuously forcing it against the heated shell. The vapors separate from the hot crude in the evaporating chamber and enter a vertical pipe which is perforated directly under the apex of each conical spreader. This pipe delivers the vapors into three outlets which again terminate on the outside in a single down pipe and lead the vapors through the upper separator *B* to the three inner chambers *L* of the recently constructed dephlegmater *D*. The inner

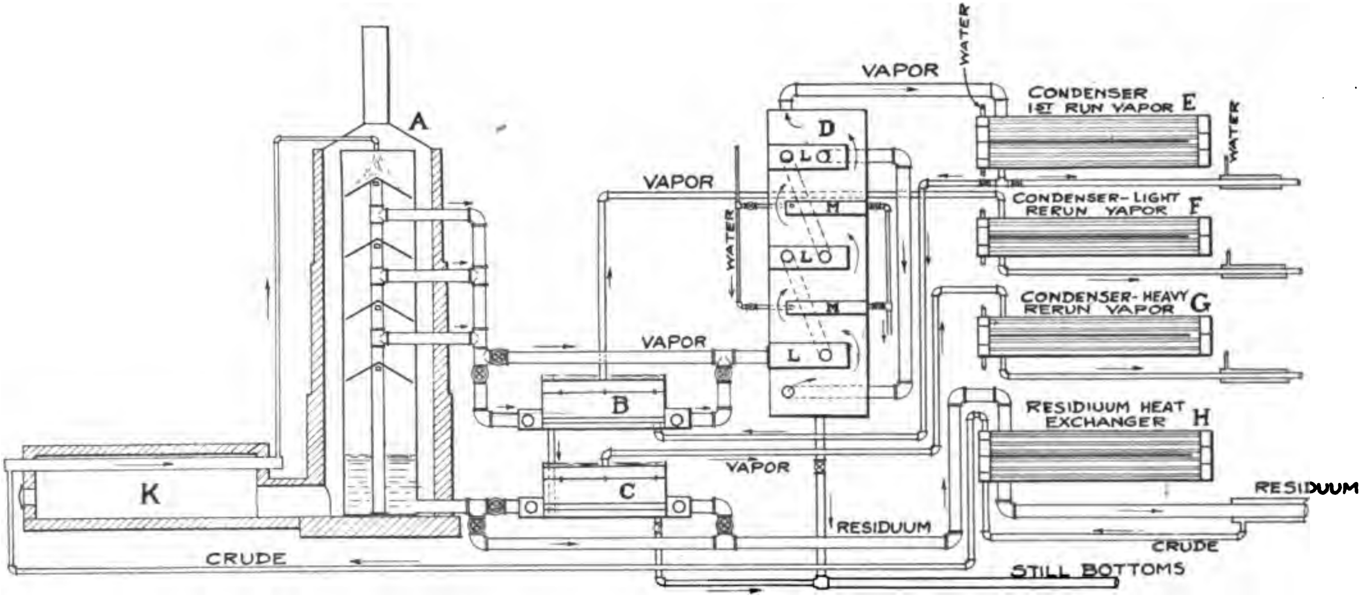
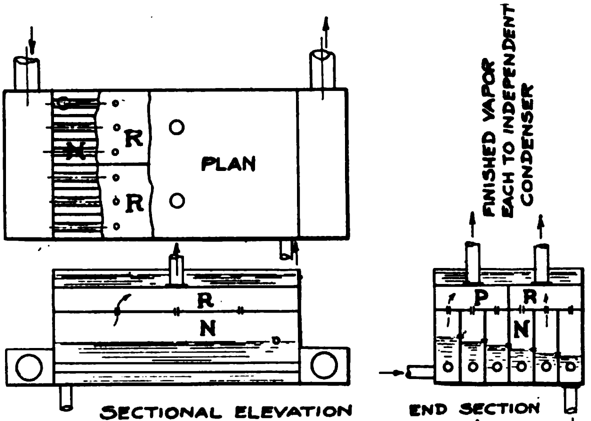

A
VAPOR
WATER
CONDENSER 1ST RUN VAPOR E
WATER
D
L
M
L
M
L
VAPOR
WATER
CONDENSER- LIGHT RERUN VAPOR F
CONDENSER- HEAVY RERUN VAPOR G
VAPOR
B
RESIDIUUM HEAT EXCHANGER H
K
C
VAPOR
RESIDUUM
RESIDUUM
CRUDE
CRUDE
STILL BOTTOMS

FIG. 19.—GENERAL PETROLEUM CO., TRUMBULL TOPPING PLANT, VERNON, LOS ANGELES COUNTY, CALIFORNIA.

chambers *L* are sealed from the rest of the interior of the rectangular box forming the dephlegmater.

Between each of the three chambers *L*, are two water compartments *M* placed across the box. The incoming vapors enter the lower chamber *L*, pass from the lower to the middle chamber *L*, and from the middle into the upper chamber *L*, and from there through a pipe into the open portion at the bottom of dephlegmater *D*. These gases coming in contact with chambers *L*, which are filled with the hotter vapors, are re-heated.

As the re-heated vapors flow up in the main box *D*, they come in contact with the cooling surfaces of the water compartment *M*, which condense and throw down the heavier distillates on to the heated surface of the chamber *L*, the latter evaporating any of the lighter oil, which might be entrained with the condensed heavier distillates. The result of this operation, it is claimed, is that all the heavy discolored distillates finally reach the bottom of the dephlegmater, while the light clean vapors pass onward and out of the dephlegmater to a battery of two large tubular water condensers *E*, which are of similar construction to the residuum heat exchangers.



SEPARATOR B & C

FIG. 19A.

The distillate from condenser *E* flows to the separator *B* entering the bottom compartment at one corner, traveling forward and backward six times through a series of compartments *N*, heated by the hot vapors coming from the evaporator *A*. (Fig. 19 A). As the distillates flow through these heated compartments *N*, they are evaporated, the vapors rising through openings into either of the two upper compartments *P* and *R*, from each of which the vapors are led to two independent condensers *F*.

The outside upper portions of the separators *B* and *C* are water jacketed, and as the hot vapors rise in the compartments *P* and *R*, any heavy distillate carried by the vapors is condensed and drops back into compartment *N*, so that only the lighter vapors escape from the respective compartments.

The unevaporated distillates pass on to separator *C*, where they are treated in the same manner as in separator *B*, the vapors being conducted to two independent condensers *G*.

On account of the distillates being heavier in separator *C* than in separator *B*, a greater heat is required to evaporate them, which is derived from the hotter residuum instead of from vapors. This arrangement of two separators with four independent compartments gives four separate re-run distillates of different gravities, practically without additional cost for fuel, the heat expended in doing this work being derived from the original heat absorbed in the retorts, except for the steam which is admitted into the separators to help drive over the vapors.

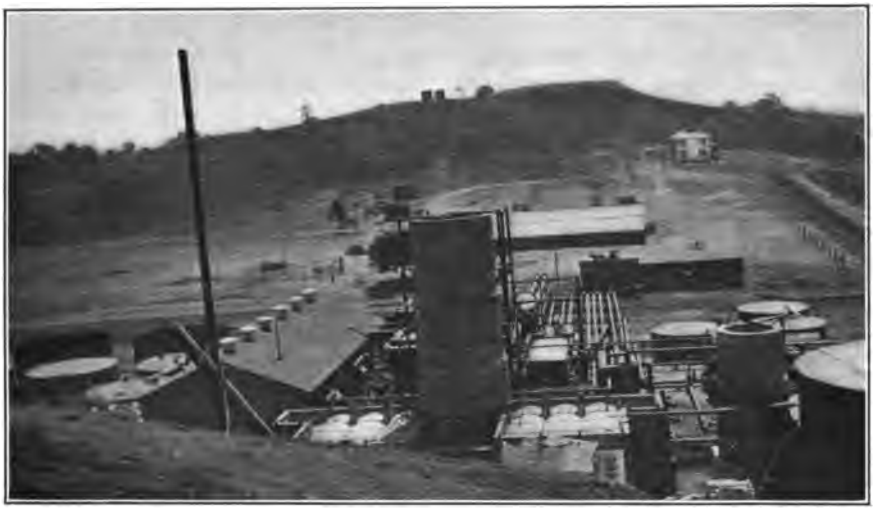


Fig. 20.—Trumbull Plant of The Shell Co.

The unevaporated distillate from separator *C* passes on with the heavy distillate from dephlegmater *D* to the storage tanks.

Bypasses are arranged in both the vapor and residuum lines, so that all or only a portion of the vapor or residuum may travel through the separators; by this means, only the required heat is obtained.

The residuum after falling to the bottom of evaporator *A*, passes through the separator *C*, and on to a series of three heat exchangers, each exchanger having a diaphragm in the shell, which guides the oil backward and forward; it then enters the next shell and so on into the third shell from which it passes through water-jacketed pipe coolers to storage.

The cold crude, entering these exchangers through the tube compartment at the end of the shell and by means of three diaphragms, is guided backward and forward through the tubes four times, this process being repeated through each of the exchangers, the crude entering the coldest exchanger and passing out of the hottest.

THE SHELL CO., OF CALIFORNIA

The Shell Company, through an affiliated company known as the Simplex Refining Co., purchased in May, 1915, all the Trumbull topping system patents, at a reported figure of $1,000,000, and is constructing at Martinez a Trumbull topping plant in connection with a complete refinery.

This plant will be supplied by an 8-in. pipe line running from the Coalinga field to Martinez, where the company has rail and deep-water facilities.

The topping plant now being constructed is a single unit and has an economical capacity of about 5,000 bbl. of crude oil in 24 hr. No money is being spared to make it as complete and economical as the process will permit. In general design, the plant is similar to the General Petroleum Co.'s Vernon plant. Many improvements have been made; one of the most noticeable is the spreading of the different parts so that they can easily be inspected and repaired; the change in design of the retorts, and various minor changes which will be to the advantage of the new plant.

Another feature of the Shell plant is the addition of an evaporator column in place of the dephlegmater lately erected at the Vernon plant.

Besides the main topping plant, the Shell company has erected a smaller Trumbull retort with its evaporating tower for the purpose of running asphaltum and doing other experimental work. This part of the plant is not complete and is not otherwise referred to in this paper.

Like the Vernon plant, all condensing and cooling will be done in tubular condensers.

The process will be carried on as follows:

The cold crude enters first the three vapor condensers *E* and *U* (Fig. 21), then passes forward and backward four times through the tubes of each exchanger, then to the coldest of the residuum heat exchangers *H*, through the five exchangers *H* in the same manner as the other exchangers, leaving the hottest exchanger to enter the top of the evaporator column *T*, which is practically of the same construction as the main evaporator except that the heat is derived from a surrounding vapor jacket. The crude on entering the column *T* is diverted to the heated side in its downward course by the conical deflectors and finally drops into the base of the column. From there, it passes through the retort *K*, then up to the top of the evaporator *A*, where in its downward course, it is again deflected against the shell heated by the furnace gases, and falls to the bottom of the tower.

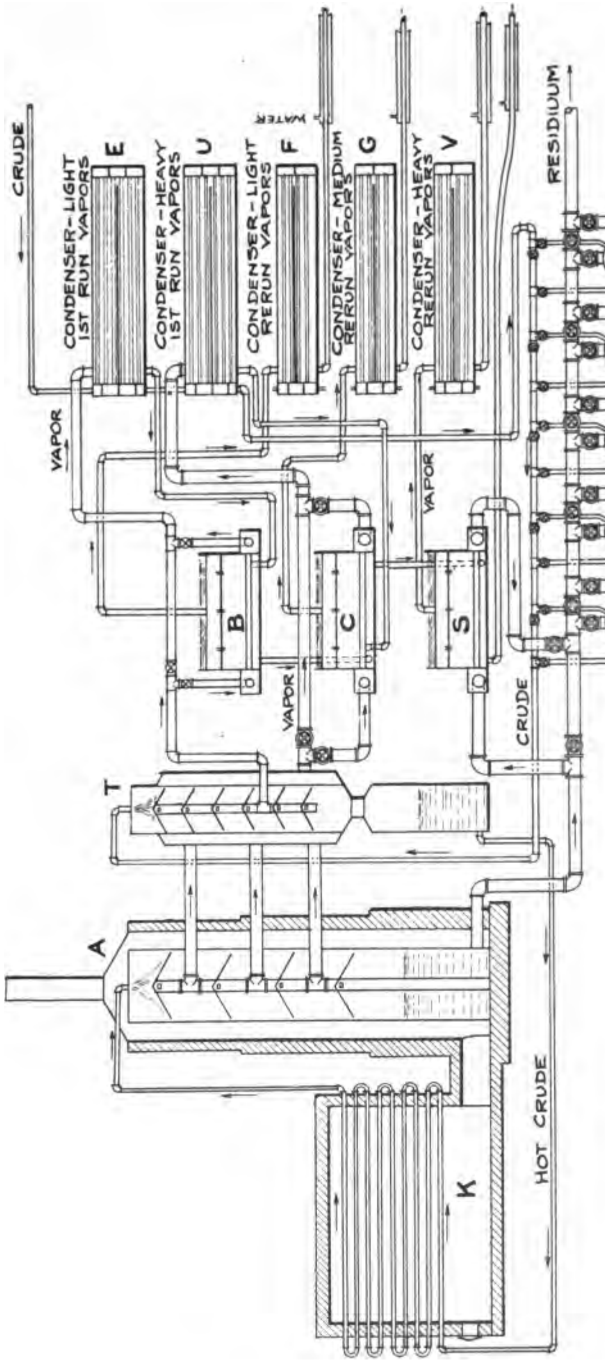

CRUDE
CONDENSER-LIGHT 1ST RUN VAPORS
E
CONDENSER-HEAVY 1ST RUN VAPORS
U
CONDENSER-LIGHT RERUN VAPORS
F
WATER
CONDENSER-MEDIUM RERUN VAPORS
G
CONDENSER-HEAVY RERUN VAPORS
V
RESIDUUM
VAPOR
B
C
VAPOR
S
VAPOR
CRUDE
H
T
A
HOT CRUDE
K

FIG. 21.—SHELL COMPANY OF CALIFORNIA, TRUMBULL TOPPING PLANT, MARTINEZ, CONTRA COSTA COUNTY, CALIFORNIA.

The first vapors to come off are from the hot crude in the evaporator column *T*, which are led to the separator *B*, passing through the heating tubes and out of the opposite end, then to the condenser *E*, where the distillates are condensed and returned to the lower compartment (*N*) of separator[2] *B*, the process of evaporation being the same as described in the General Petroleum Co.'s Trumbull plant.

The vapors on leaving separator *B*, which will have the lightest gravity, are condensed in condenser *F* and then pass off through jacketed pipe coolers to storage, while the unevaporated distillates from separator *B* pass on to separator *C*.

The second vapors coming off from evaporator *A* are collected in the central perforated pipe and led to the jacket of the evaporator column *T*, after imparting a certain amount of heat to the inner shell, they pass through the heating tubes in separator *C*, then to condenser *U*, the condensed distillates being returned to compartment *N* of separator *C* to be evaporated as previously described. The vapors then flow to condenser *G*. These distillates are of a heavier grade than those from separator *B*. The unevaporated distillates flow from separator *C* to separator *S*, which is heated by the hot residuum passing through it on its way to the exchanger.

The vapors from separator *S* pass off to the condenser *V* in the same manner as those from *B* and *C*; they are heavier than those from the other two separators. The unevaporated distillate from separator *S* is not further refined and is run to storage.

Each of the three separators has two separate vapor chambers, each having a separate outlet connected to its independent condenser so that the three separators will give six different cuts of varying gravity of rerun products. An additional product will be the still bottoms drawn off from the separator *S*.

The Shell Co. is designing an additional Trumbull plant of equal capacity to be erected alongside of the present one, in which still further improvements are contemplated.

ASSOCIATED OIL CO.

Plant at Gaviota, Santa Barbara County

The Associated Oil Co. has a topping plant of about 5,000 bbl. daily capacity at Gaviota, Santa Barbara County, which consists of two systems; one a battery of ordinary stills and the other a battery of pipe retorts. As I have not dealt in this paper with ordinary stills, but have confined myself entirely to the other types, and as the Associated Oil Co. is building a much larger pipe retort topping plant at its Avon refinery, of

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[2] For detail drawing of separator see Fig. 19A.

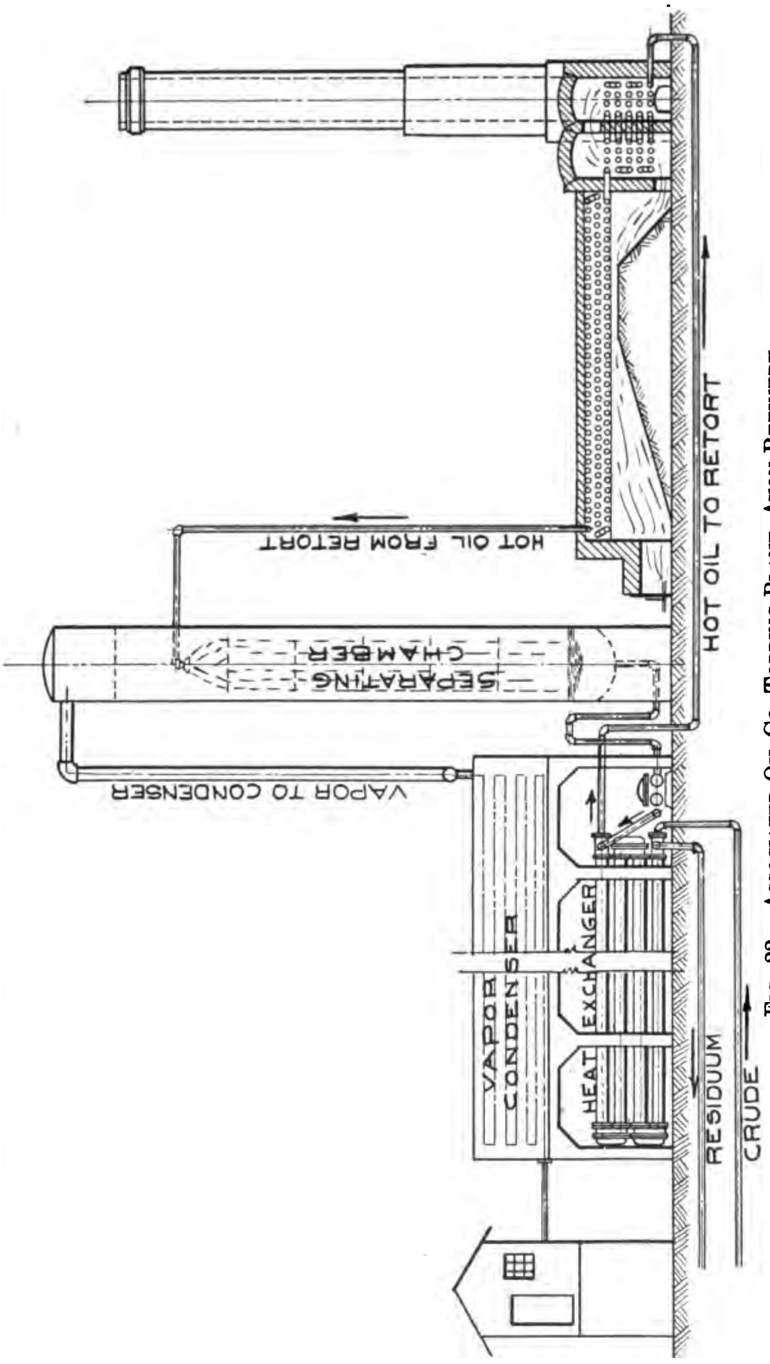

FIG. 22.—ASSOCIATED OIL CO. TOPPING PLANT, AVON REFINERY.

the same general design as the one at Gaviota, I will not describe either of the Gaviota systems but will describe the topping plant now being constructed at the Associated Oil Company's new refinery at Avon.

Plant at Avon, Contra Costa County

A retort topping plant is being erected as an addition to the existing refinery and consists of a flat square construction, divided into three separate furnaces. The direction of all flame is at right angles to the pipes. Placed above the three furnaces will be three rows each of 34 4-in. pipes, about 20 ft. in length with flanged return bends at either end. At the rear end of the furnaces is an enlarged flue which contains 40 4-in. pipes for the purpose of absorbing as much heat from the flue gas as is practicable. By constructing three long furnaces, over which the pipes are placed at right angles, the pipes can be supported at four points. Also, short cast-iron T-bars are placed between each row of pipes over the division furnace walls, which support the upper rows and prevent bending of the pipe due to the action of heat. The roof over the furnaces, which is covered by a double thickness of hollow tile with a layer of gravel, rests on the upper row of pipes.

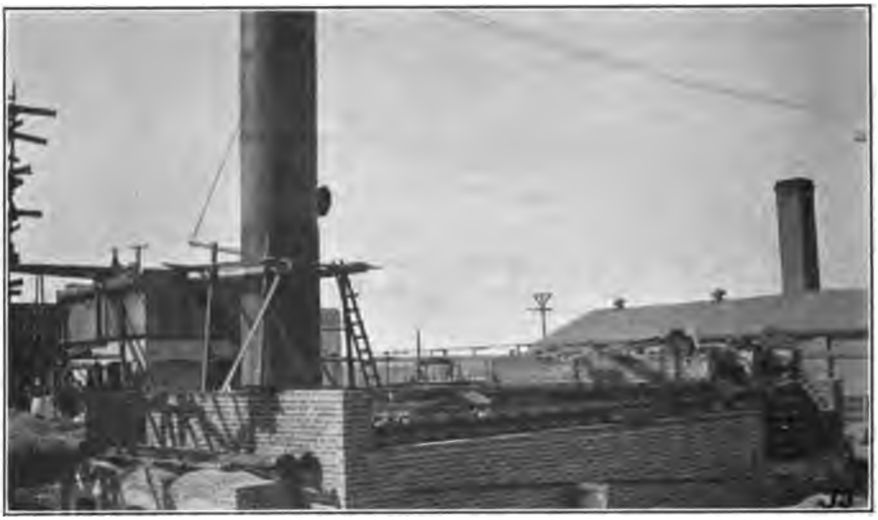

FIG. 23.—VIEW SHOWING FRONT OF FURNACES UNDER CONSTRUCTION.

The main outside walls of the furnace come just inside of the return bends at either end of the pipe. There is also a light auxiliary wall placed on the outside of the return bends. The space between the main wall and this auxiliary wall is covered on top with short light sheet-iron plates which prevent a circulation of cooling air on the return bends. The whole construction admits of removing any pipe or joint without loss of time.

In connection with the retort is a separating tower about 5 ft. in diameter and 40 ft. high, into which the heated residuum enters at a point about 10 ft. from the top. The residuum strikes a succession of baffles as it falls to the bottom of the separator; the light vapors flow upward and out of the vent pipe at the top of the separator.

The heat exchanger consists of a nest of 24 10-in. pipes about 40 ft. long mounted on a steel frame, the 10-in. pipes being being fitted with return bends, and having five 2-in. pipes passing through each from end to end with connections so arranged at either end of the 10-in. pipes that the heated residuum begins its flow at one end of the 10-in. pipes and flows backward and forward through each successive 10-in. pipe until it has reached the end; in the opposite direction the cold crude oil is admitted into the 2-in. pipes at the cool end of the exchanger and leaves the exchanger at its hottest end.

An ordinary water-cooled condenser for condensing the vapors from the separator is used in connection with the retort.

In operating the plant, the crude oil enters the exchanger, passes out of the hottest end and enters the retort, at a pressure of about 50 lb., through the lower corner pipe of the flue chamber, traverses backward and forward through the eight lower pipes in this chamber, then flows successively through each row upwards to the top; from there it traverses backward and forward over the bottom row of pipes lying over the furnace until it reaches the front, then in the same manner through the second and third rows back to the front of the furnace; from there it is led, by a pipe with a pressure regulating valve in it, to the upper portion of the separator where the spraying takes place. The residuum is then pumped from the bottom of the separator through the heat exchanger to the storage tanks. The vapors from the separator are led direct to the condenser.

The residuum will be delivered from the retorts to the separator at about 450° F. As this particular topping plant has not yet been completed, I cannot give definite data as to its operation, but the same design of exchanger used in connection with ordinary stills topping oil in the same plant, receives the residuum at 450° F.; the crude oil enters the exchanger at 70°, leaves it at 260°, and enters the stills at approximately that temperature.

The estimated economical capacity of this plant with clean oil will be about 5,000 bbl. of crude each 24 hr., taking off about 15 per cent. of 50° Bé. tops.

In closing this paper, I wish to express my appreciation of the courtesies extended to me by the officers and employees of all the companies herein referred to, as it would have been impossible to compile so much accurate data without their hearty cooperation.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Geology of the Iron-Ore Deposits In and Near Daiquiri, Cuba[*]

BY JAMES F. KEMP,[†] NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

THE iron ores of southeastern Cuba present a subject of exceptional geological interest. Their relations to the inclosing rocks are varied and in some cases unusual. The problem of their geological origin is not a simple one and for this reason has occasioned some divergence of views among previous observers. Profiting, however, by these earlier studies and by more detailed field and laboratory observations as well as by the experience of added years of mining, much more satisfactory descriptions and interpretations can now be prepared.

PREVIOUS WORK

In 1884, interest in the southeastern coast of Cuba became active among the consumers of iron ore along the Atlantic seaboard of the United States. Active development was undertaken by the Juragua Iron Co., Ltd., a corporation affiliated with the Bethlehem Steel Co., and the Pennsylvania Steel Co. The property of the Juragua company, which is situated about 15 miles east of Santiago, was studied in June and July, 1884, by Dr. James P. Kimball, at the time Professor of Economic Geology in Lehigh University and one of the best known of American mining geologists. Dr. Kimball published on his return two papers, the first important descriptions which we have of the local geology.[1] It is indeed true that 28 years earlier, D. T. Ansted,[2] an eminent English geologist, had recorded an account of the copper deposits of El Cobre, *west* of Santiago, and of others near Cienfuegos. From the earlier contribution one may glean a few details of the local geology at these two mines, but to Dr. Kimball we owe the first easily accessible descriptions of the region to the *east* of Santiago.

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[*] In the preparation of the present paper the writer has had the cordial support and interest of Charles F. Rand, President of the Spanish-American Iron Co. For this aid the fullest acknowledgment is here gladly made.

[†] Professor of Geology, Columbia University.

[1] James P. Kimball. Geological Relations of the Specular Iron-Ores of Santiago de Cuba, *American Journal of Science*, vol. xxviii, third series, pp. 416 to 429 (1884). The Iron Ore Range of the Santiago District of Cuba, *Trans*. xiii, 613 to 634 (1884–85). To the combined reprints of the above papers as distributed to his friends, Dr. Kimball added reproductions of three maps which accompanied his report to his clients. The first is of the East and West Mines near Firmeza, on a scale of about 150 ft. to the inch and with contours at 3-m. intervals. Some notes on the geological formations are written on the map. The second contains three separate figures, viz.: (1) an outline map of the coastal district from the Bay of Santiago to the Rio Bucano; (2) a geological section through the Juragua Hills to the Gran Piedra; (3) a claim map of the surroundings of Firmeza. The last plate contained a series of geological sections and panoramic sketches of the mines and orebodies near Firmeza.

Dr. Kimball describes a syenite as the basal and oldest rock along the coast. He saw its outcroppings in a belt near the sea and learned of its supposed reappearance in the higher portions of the mountain range which culminates in the Gran Piedra. He therefore inferred that it existed beneath the diorites which form the country rock of the intervening belt and which contain the iron ores. Near the sea Dr. Kimball observed the coral reefs which had been obviously raised above the water in recent geological time and which stood at three elevations, respectively 14 ft., 175 ft., and 350 ft. above tide. Farther back the diorite was believed to cover the syenite, through which in some places it may be seen coming up in dikes. The diorite was believed to be thinnest toward the coast and to reach a maximum of about 2,000 ft. farther back. Dr. Kimball inferred that the diorite welled up as an outbreak of igneous rock through the syenite and that from sources of supply back toward the Sierra Maestra it flowed southward toward the sea. The flood of igneous rock was either checked by the salt water or failed in amount and so covered no greater area than approximately that which we now observe. In its course the diorite was supposed to overwhelm reefs of coral limestone, perched on the syenite and still older than those now visible. The effect of the diorite was to turn these older reefs into the crystalline marble which may sometimes be seen in association with the orebodies. Subsequently the circulating rain-waters, acting on the diorites, which contain much iron oxide, took the iron oxides into solution and encountering the ancient coral reefs, were relieved of their dissolved burden so as to replace the limestone with the ore. Dr. Kimball observed structures still preserved in the ore which suggested the organic growth of coral.[3]

These observations are of great interest, but so far as the origin of the ore is concerned are chiefly of value in bringing out the probable precipitation of some of the ore, perhaps indeed of most of it, through the agency of limestone. The precipitating limestone was not an old coral reef perched on the syenite, but a member of an important sedimentary series which is probably the oldest visible rock in the region.

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[2] D. T. Ansted. The Copper Lode of Santiago in Cuba, *Quarterly Journal of the Geological Society of London*, vol. xii, pp. 144 to 153 (1856). Extended quotations from this paper are given by Hayes, Vaughan, and Spencer, *Report on a Geological Reconnaissance of Cuba*, pp. 44 to 50 (1901).

On the San Fernando Copper Lodes near Cienfuegos in Cuba, *Idem*, vol. xiii, pp. 240 to 242 (1857).

[3] The writer has also observed similar cases at Sigua and has been reminded of the supposed fossil coral of the pre-Cambrian rocks of Quebec, the Eozoon Canadense, which was an object of so much study in former years, but which is no longer regarded as of organic origin.

The next paper upon the district is by F. F. Chisholm,[4] who specially examined the Berraco group some six years after Dr. Kimball's studies, and who also visited the Juragua company's mines. Mr. Chisholm reviews the relations of the syenite and diorite, and notes the three terraces of recently uplifted coral. With the view, however, that the ore has replaced ancient coral reefs he dissents entirely. Much better exposures had been prepared at Firmeza in the Union mine during six years of work. A vertical body 200 ft. high and of goodly although not definitely stated width had been exposed. Quite naturally Mr. Chisholm found difficulty in believing that it was an old coral reef replaced with ore. The slabs of limestone observed in the diorite near the ore were interpreted as fragments torn off by the diorite in its upward journey and derived from some older formation. The great vertical orebody at the Union mine was inclosed in diorite. It had a wavy band of chloritic clay on the east side of the ore, with small parallel, lenticular streaks of vitreous quartz. Outside the clay was 2 ft. of sand, loosely compacted and colored by manganese, but displaying a sharp contact against the diorite. The other openings at Juragua were similar but presented less clearly marked relations. At Berraco, by way of contrast, no limestone was exposed. Mr. Chisholm finally interpreted the ore as a direct concentration in an igneous dike, which was originally characterized by a high percentage of iron; or else it is a distinct band forming a portion of a larger dike. The source of the ore would therefore be below, and the ore itself would continue below the effects of atmospheric action.

Two years after Mr. Chisholm's visit, an examination of the Sigua group together with visits at Juragua and Daiquiri was made by the late George W. Goetz of Milwaukee, whose observations were communicated to the late Professor Hermann Wedding of Berlin, partly by letter and partly through short articles in the press. Professor Wedding assembled the data in a paper although not himself personally familiar with the mines.[5] The oldest rock is stated to be the syenite of earlier writers, upon which rests an oolitic, coralline limestone belonging to the Jurassic and of the horizon of Quenstedt's Beta, of the upper White Jura. The limestone is much disturbed and together with the syenite is pierced by outbreaks of diorite, which have changed the limestones neighboring to them into white marble. The red hematite is associated with the diorite in the most intimate manner as in most localities the world

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[4] F. F. Chisholm. Iron-ore Beds at the Province of Santiago, Cuba. *Proceedings of the Colorado Scientific Society*, vol. iii, p. 259 (1888–90).

[5] H. Wedding. Die Eisenerze der Insel Cuba, *Stahl und Eisen*, vol. xii, No. 12, pp. 545 to 550 (June 15, 1892).

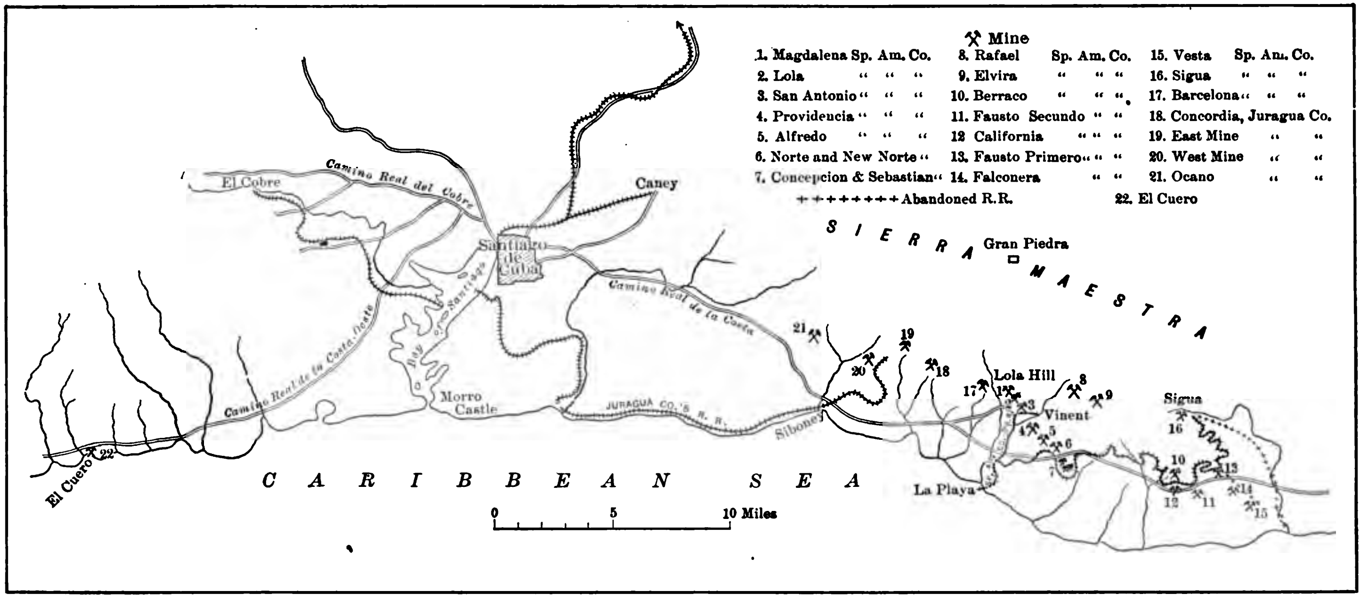

FIG. 1.—SKETCH MAP OF THE MINES EAST AND WEST OF SANTIAGO DE CUBA.

over, such as the Lahn district in Germany, and the Lake Superior district in North America. West from Santiago Devonian sandstone and Silurian limestone constitute the stratified rocks. Dr. Wedding emphasizes the great abundance of surface blocks of ore in the iron districts east of Santiago, and thinks it probable on the analogies with other districts that the iron ores form beds rather than veins. The exposures described as veins may be folded beds. The diorite is often porphyritic and the iron ore is at times involved with chlorite and epidote; and frequently appears simply as a replacement (pseudomorphose) of limestone. Many details of the engineering features follow.

In the interesting work, *Cuba and Porto Rico, with the Other Islands of the West Indies* by Robert T. Hill,[6] a very brief sketch of the geology of Cuba appears on pp. 40 and 41, together with a very small geological map after one by de Castro and Salterain. Limestones of Eocene-Oligocene age constitute the coast east and west of Santiago, with an intrusive mass of granitoid rock at Daiquiri.

The district received careful study from Dr. A. C. Spencer in connection with the reconnaissance of Cuba carried out under General Leonard Wood's direction, during the American occupation in 1901.[7] Dr. Spencer gives an excellent historical sketch of the development of the enterprises and, regarding the engineering features, quotes extensively from a report by Charles M. Dobson, E. M., to the Military Government, a report which must have been made shortly before the date of his own report. Dr. Spencer determined the syenite of earlier writers to be a coarse quartz-diorite, whose exposures are well shown from Siboney to the Playa, the port of the Daiquiri district. It is covered, in the hills near the shore, by a thin veneer of coral rock. The quartz-diorite (sometimes called merely diorite by Dr. Spencer) extends about three miles inland and is succeeded by a belt or zone of intrusive masses of porphyry. "Near the southern edge of this zone the porphyry occurs in extremely irregular dikes and these dikes become more numerous in the higher foothills to the north. It is within these foothills that the iron ore occurs. Above them the slopes of the Sierra Maestra are composed of bedded volcanic rocks with a few intercalated beds of limestone or limestone breccia. The lower flows are of a basaltic nature, but some of the uppermost are rhyolitic, while upon the top of the mountain the massive rocks are overlaid by rhyolitic breccias, which constitute the base of a series of volcanic rocks occupying the northern slope of the range. These features are indicated in the accompanying diagram Fig. 2. (Fig. 9 of Spencer) which represents also the general geology and structure of the Sierra Maestra from Cabo Cruz to the vicinity of Guantanamo Bay."

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[6] The Century Co., New York (1898).

[7] C. W. Hayes, T. W. Vaughan and A. C. Spencer. *Report on a Geological Reconnaissance of Cuba,* with a geological section from the coast to the summit of the Sierra Maestra and four small sections of the orebodies, pp. 69 to 82 (1901).

The volcanic rocks are not intimately associated with the iron ores so that the chief interest attaches to the porphyry, which is the main wall rock of the ore. Dr. Spencer was impressed by the association of hornblende schist as well as crystalline limestone with the ore as exposed in the mines of the Juragua Company, but not much in evidence in the workings on Lola Hill of the Spanish-American company. He was led by these observations and by others elsewhere in Cuba to infer the existence of an old metamorphic series, even older than the serpentine so abundant in some parts of the island. The outbreaks of the quartz-diorite and the porphyry preceded the Oliogocene epoch of the Tertiary period, since younger beds on the northern slope of the Sierra Maestra contain fossils of this time. After discussing, at too great length to be cited verbatim here, the derivation of the ores by possible magmatic processes in the igneous mass, and by replacement processes of the included schists and limestones, Dr. Spencer favors the view that the ores were original members of the schist and limestone series and that they have been torn off with varying amounts of their old associated strata and floated upward as included masses in the porphyry. The great slabs or masses of ore are often cracked apart and separated by dikes of porphyry which are regarded as the same rock with the general inclosing walls.[8] Several cross-sections assembled and reproduced here in Figs. 3, 4, 5 and 6, on a reduced scale are given in support of the view. One of

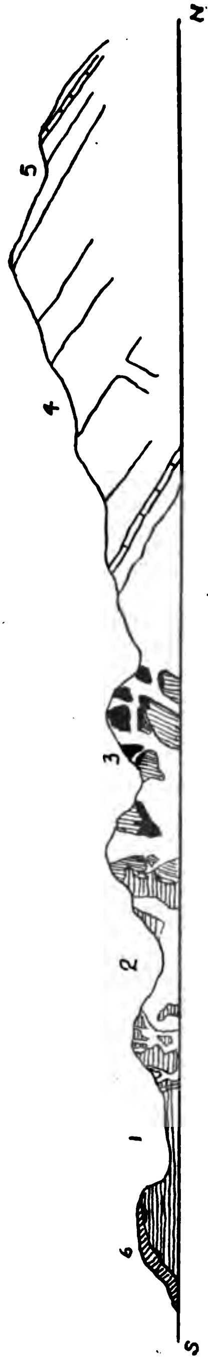

FIG. 2.—DIAGRAMMATIC GEOLOGIC SECTION THROUGH IRON-ORE ZONE FROM THE SOUTH COAST TO THE SIERRA MAESTRA.

1. Diorite of Spencer, called granite in this paper.
2. Diorite porphyry, here called diorite.
3. Iron ore and inclosing schists.
4. Basalt tuffs and limestone breccia.
5. Coarse, acid, volcanic breccia.
6. Coral rock.

(Reproduced from Fig. 9, opposite p. 78, of Report by A. C. Spencer, and slightly reduced.)

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[8] In *Trans.*, xlii, 167, 168 (1911), two panoramic views appear accompanying some notes incidental to the Canal Zone excursion.

these with a quotation appears also in J. E. Spurr's *Geology Applied to Mining,* p. 118 (1907) where it is more generally accessible than in the original paper.

In reading Dr. Spencer's description one must bear in mind that the rock called syenite by earlier writers is named by him quartz-diorite or simply diorite; whereas the rock called diorite by earlier writers and appearing as the wall rock of the ores is called by him porphyry. Dr. Spencer's quartz-diorite or diorite must not be confounded with the diorite of the others.

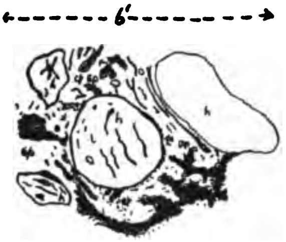

FIG. 3.

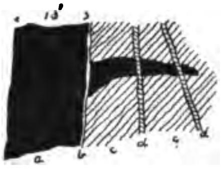

FIG. 4.

FIG. 3.—MAGNETITE AND EPIDOTE NEAR OREBODY IN EAST MINE, FIRMEZA. *h,* Hornblende schist; *ep,* granular epidote. (Reproduced from Fig. 10, p. 80, of Dr. Spencer's Report.)

FIG. 4.—SECTION THROUGH SMALL OREBODY IN NORTHEAST MINE OF JURAGUA COMPANY. *a,* Magnetite; *b,* selvage impregnated with chalocopyrite; *c,* white marble belonging to the schist series; *d,* fine-grained porphyry. (Reproduced from Fig. 11, p. 80, of Dr. Spencer's Report.)

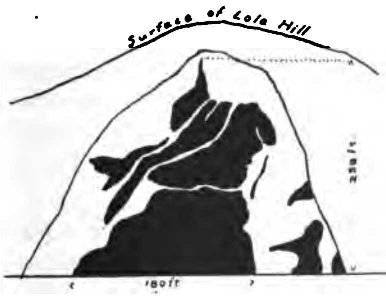

FIG. 5.

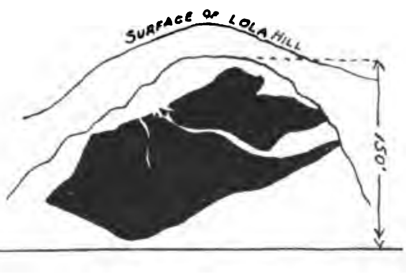

FIG. 6.

FIG. 5 (FIG. 12, *idem*).—WORKING FACE OF LOLA MINE, SHOWING GENERAL RELATIONS OF OREBODY TO INCLOSING PORPHYRY.

FIG. 6 (FIG. 13, *idem*).—GENERALIZED VERTICAL PROJECTION OF MAGDALENA OREBODY AT DAIQUIRI TO ILLUSTRATE INTRUSION OF IGNEOUS ROCK.

So far as the sedimentary rocks and especially the limestones are concerned, Dr. Kimball regarded them as coral reefs older than the modern reefs, but although no definite geological epoch is assigned to them, apparently in his view they grew in conditions not greatly changed from those of today except in the matter of subsidence. Professor Hill quoting two Spanish geologists refers them to the Eocene-Oligocene, but Dr. Spencer believes them older than the Oligocene because on the north side of the Sierra Maestra, Oligocene fossiliferous strata rest upon the same types of eruptives as appear on the south side and are later. Mr. Goetz must have supplied Professor Wedding with some sort of palaeontological proof for making so close a determination of them as a subdivision of one portion of the Jurassic. Fossils would be of extreme interest. One cannot well avoid suspecting that they are a continuation of the thick and tilted limestones which one traverses along the wonderful mountain highway, which climbs the escarpment northwest of Santiago to 2,000 ft. above the sea. The winding road crosses in true Alpine manner a great section of well stratified limestones with a northerly dip of 30 degrees or more.

In detailed areal work in the Daiquiri region as soon as one leaves the artificial exposures of the mines and railway cuts, or the freshly eroded ledges of the brooks, the difficulties of the tropics assert themselves. The weathering is so severe and the advance of decomposition so rapid, that fresh rock is often hard to find. Kaolinized and chloritized specimens do not always give a satisfactory indication of the nature of the eruptive rocks. By the aid, however, of E. W. Kohl, Jr., the engineer of the Spanish-American Iron Co., and his assistant, Alfred H. Weaver, a carefully located and quite complete illustrative series of rock specimens, over 800 in number, were collected from the lands of the Spanish-American Iron Co. The general geology has been worked out with the aid of these in the areas not actually visited. Slides for microscopic study have been freely prepared. The writer has also visited the larger mines of the Juragua company and has been further aided by a collection of specimens of ores and rocks made in 1913 by C. R. Corning, to whom acknowledgments are due.

GEOGRAPHICAL RÉSUMÉ

In connection with the iron ores we are deeply concerned with a portion of Cuba which lies east of Santiago along the coast of the Caribbean Sea. The iron mines are limited to a belt which extends not over four or five miles back from the shore. The mines have not as yet been developed more than 20 miles east of Santiago Bay nor nearer than five miles to it. They do not come down to the shore but are first found about a mile and a half from the salt water. They are thus limited to an east and west belt two to three miles wide. The ores are in an exceedingly steep series of hills whose slopes at the larger mines approximate 35 degrees, the angle of repose for loose materials. Some smaller openings are at elevations less than 500 ft. above tide, but the larger ore bodies outcropped originally up to and slightly above 1,000 ft., and cut through the high, steep hills or ridges, in actual or practically vertical position. The smaller orebodies are of all sorts of shapes and are irregular in their distribution.

To the west of Santiago several enterprises have been started, of which at present only the one at El Cuero, eight miles from Santiago Bay is active. Its ores lie quite near the shore and are reported as similar in geological relations to those on the east.

Thirty-seven miles westward from the mouth of Santiago Bay is Guama, with a port called Chirivico. The Cuban Steel Ore Co. made extensive developments on some iron-ore lands about 14 years ago, but the enterprise is not now active.

Outline of the Geology at Daiquiri

The oldest geological formation on the lands of the Spanish-American Iron Co., so far as our observations go, is of sedimentary character. Its chief representative today is limestone, but quartzites have also been discovered and at Sigua black slates are to be seen in close association with the limestone. This series is cut up into small irregular exposures by the extensive development of igneous rocks which have greatly metamorphosed it, and have produced contact zones of marked perfection and scientific interest.

The limestone can be seen in the Barcelona claim in a not appreciably metamorphosed condition. It is then a dense blue variety, strongly reminiscent of the Cambrian and Ordovician limestones of the Appalachian area of the United States, and the great Cretaceous limestones of northeastern Mexico. Satisfactory dip and strike were not easy to secure on account of the massive character and the neighboring igneous rock. One reading on the Barcelona claim was N 55 W., 53 W.. The limestone is again abundant to the north of Vinent, and to the northeast toward Sigua. It has been found on the south at various points even so far to the east as the Coco River. While we have no positive evidence that all the exposures belong to the same formation, yet in the absence of the evidence of fossils, no one of which has been yet discovered, we may assume that they do.

Quartzites have been discovered to the east of La Playa, the port of the Daiquiri district, and extend nearly to the Coco River in an east and west belt from a half mile to a mile and a half back from the shore. They are chiefly quartz sandstones of pronounced fragmental character but show the effects of metamorphism. Sericite is recognizable and apparently zoisite. There are particles of feldspar. Fig. 7 will illustrate them.

In the pits at Sigua a fine dark-green slate was found, which under the microscope is a greenish, almost isotropic mass with a few grains of quartz. It was interpreted as a fine feldspathic tuff, now kaolinized and stained green with chlorite.

There are other fragmental rocks both near the mines at Sigua and along the trail to the southeast. They were once feldspathic in composition but are now so greatly kaolinized as not to be more sharply determined. They seemed to be an old volcanic breccia. The strike and dip some two or three miles southeast of Sigua were N. 20 E., 10 W. We have no evidence of the relations of this rock to the limestones and quartzites.

It is possible that in part or as a whole these sediments may represent the ancient metamorphic series mentioned by Dr. Spencer. But no hornblende schist has been found away from the mines, and such apparently schistose representatives as were observed in the Spanish-American company's workings were lime-silicates, obviously the results of contact metamorphism. Abundant evidence of faulting movements and of shearing under pressure are to be noted both in the open cuts of Lola Hill and near Firmeza. The limestone-quartzite series does not seem to be so much metamorphosed as would be implied in Dr. Spencer's description of the ancient schists.

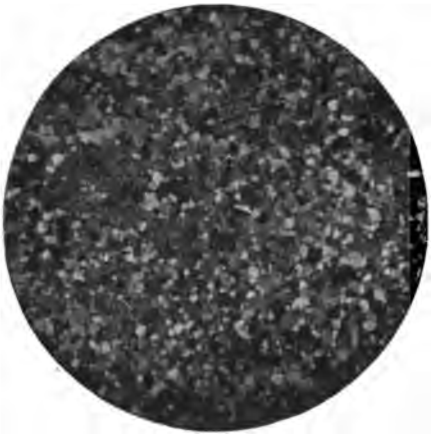

FIG. 7.—QUARZTITE EAST OF LA PLAYA.

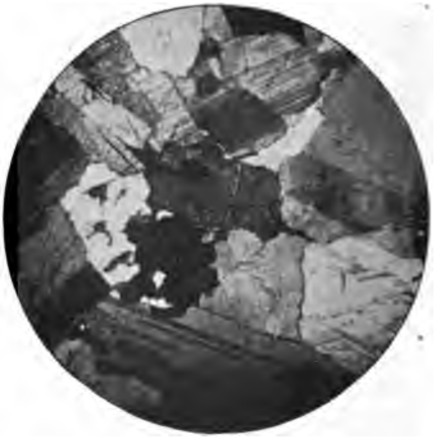

FIG. 8.—GRANITE (QUARTZ DIORITE), BERRACO R.R.

In FIG. 7 and in all the subsequent reproductions of thin sections (except Fig. 43), the actual field is 0.1 in. or 2.5 mm.

Granite.—The review of the earlier papers has shown that there has been a general disposition to regard the syenite or quartz-diorite as the oldest rock in the region. As the writer is anxious to avoid all confusion of this rock with the diorites which are associated with the ore, it will be called granite. The ordinary field worker would unquestionably collect it as granite. When examined under the microscope it always has abundant quartz, making syenite an undesirable name. Sometimes the feldspar is predominantly orthoclase; sometimes, and more often, it is oligoclase. Biotite and hornblende and augite are all three to be seen in one slide or another but biotite is most common. The usual zircons, titanites and magnetites appear and at times there is a marked tendency to develop micro-pegmatites. The rock is rather coarsely granitoid in texture and of light or actually white color. Dr. Spencer's description of the rock as quartz-diorite coincides perfectly with most of the thin sections examined by the writer. These are, however, phases well over toward granite, which name, both from ordinary field usage and to avoid confusion is here employed. We know the granite to be later than the limestones because it changes the latter to beautiful white marble and in places has produced contact zones of great perfection. Along the Berraco R. R., where it skirts the east side of the San Sebastian claim there is one contact zone of coarsely crystalline epidote right between the granite and the white marble. The ore of the San Sebastian is in a garnet zone produced by the granite from the limestone; while that of the neighboring Concepion claim is in epidote produced in the same way. There can be no question that the granite is intrusive in the limestones and is later in age. But the granite itself is at times abundantly provided with angular inclusions of a much darker diorite. In the cuts along the Berraco railway along the east side of the San Sebastian claim, the ledges of granite display a good many of these angular fragments and lead one to believe in some older unidentified member of the composition of diorite. On the other hand, and as all the previous observers have mentioned, dikes of diorite cut across the granite and are generally believed to be the supply conduits for the great masses of diorite which contain the ore. Yet there are dikes in the mines which are also later than the diorite including the ore and which may be of the same group as the dikes that are to be seen cutting the granite. No positive evidence has been obtained by the writer to show that the granite is older than the ore-bearing diorite. The included fragments which appear in the granite may have been derived from the latter rock, unless the earlier observations are destructive of this view. As will be later shown granitic, pegmatitic and even dacite-porphyry dikes cut the orebodies in one mine and another. Should they be offshoots from the deeper-seated granite, the latter might even be later than the main orebodies themselves. The included, angular fragments would thus correspond to the older diorite which contains the orebodies.

The granite is best developed east and west of La Playa, from which it extends nearly to Vinent. It is also seen in good exposures along the railway from Firmeza to Siboney. Undoubtedly it extends some miles along the coast, Siboney to La Playa and beyond each.

Diorite.—In the neighborhood of Vinent the granite yields to a rather coarsely crystalline diorite which extends some distance to the north and is widespread east and west. The diorite is sometimes visibly porphyritic from phenocrysts of feldspar and for this reason was probably called porphyry by Dr. Spencer. On careful microscopic study the writer is unable to draw any very sharp distinctions as against the wall rocks of the orebodies and therefore is disposed to regard this great dioritic intrusive as essentially one geologic unit.

The diorite is a gray rock, which may assume a light color on weathering, but which, in the varieties rich in hornblende may be a very dark gray. In some exposures it is of moderately coarse grain, comparable with an average granite having components up to 0.2 in. or 5 mm.; again it is much finer. The most abundant mineral is plagioclase, well twinned and affording extinctions on either side of the plane of twinning from a few degrees to as high as 25. Varieties in the andesine and labradorite series are thus chiefly represented with an occasional member as basic as bytownite. In the coarser varieties the plagioclase has the usual habit of the components of the granitoid rocks. The crystals are roughly as long as broad and of irregular outline. In the more finely crystalline phases there is a marked tendency of the plagioclase to become rectangular and even at times square, reminding one of the characters of the dike rocks. Yet the specimens were certainly gathered from the large intrusive mass as distinguished from the relatively narrow dikes. The plagioclases are rarely zonal. Untwinned feldspar is only now and then to be seen, so that orthoclase is a minor component. Quartz is not uncommon, but it is only a subordinate member. The feldspar may be four-fifths the slide as a maximum and one-half as a minimum. The hornblende is the common green variety in almost all the slides, but it shows frequent indications of having once been brown and to have bleached in alteration. One constantly suspects common green hornblende of having been derived from augite, but the probabilities are very strong that in the Daiquiri rocks the hornblende is an original mineral. In the coarser diorites it is irregular in outline but in the finer-grained varieties in which the plagioclase becomes rectangular, the hornblende exhibits excellent prismatic development. Hexagonal sections and elongated ones with parallel sides are the rule. The hornblende may make up as much as one-third the section; it may sink to a fifth. Irregular magnetite is freely associated with the hornblende, but seldom appears in the feldspar. Apatite is often seen and is even included in the magnetite, being thus an older mineral. Pyrite is now and then an associate of the magnetite.

In some slides and even those in which alteration has not made particularly great progress, calcite may nevertheless be detected. It is secondary and has undoubtedly been yielded by hornblende and feldspar. From this outline, which may be considered the normal, variations occasionally appear. Biotite in a visibly bleached or discolored form is not infrequent as a minor associate of hornblende, but it has never been observed in more than a subordinate rôle. In an abandoned cut in the southern workings of the old El Norte mine a specimen of diorite was gathered which has much colorless augite associated with the hornblende. The plagioclase at the same time becomes basic. Its extinctions reach 25 degrees with the twinning plane. Epidote has also begun to develop.

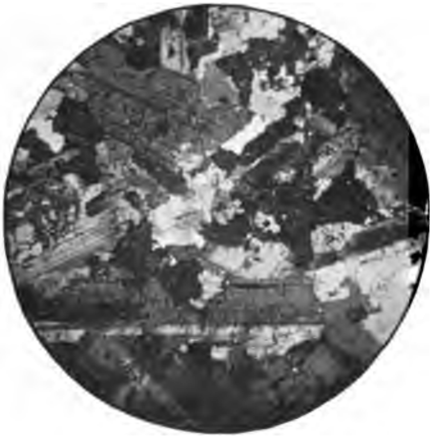

Fig. 9.

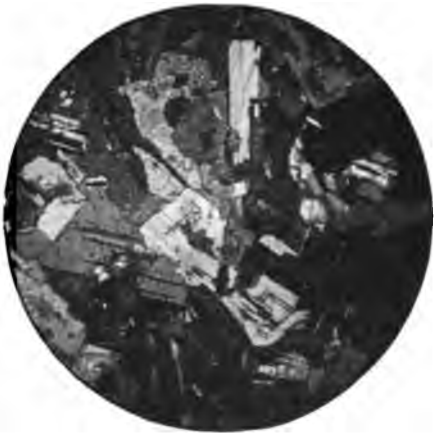

Fig. 10.

Fig. 9.—Diorite, Lower Tunnel, El Norte Mine.

The mineral with parallel bands is plagioclase. The clear white is quartz. The dark mineral is chiefly hornblende with a little magnetite. Crossed nicols.

Fig. 10.—Diorite from East Wall of East Mine, Firmeza.

The slide is chiefly plagioclase. Hornblende is recognizable in the upper middle part by its cleavage. The very dark minerals on the right lower side are extinguished plagioclase. The rock is practically identical with the one illustrated in Fig. 9.

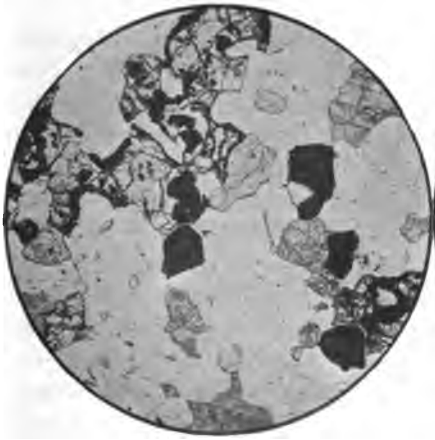

Fig. 11.

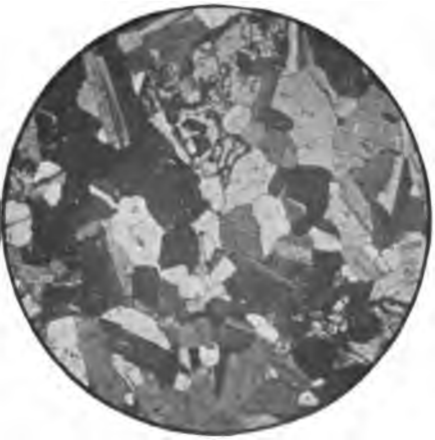

Fig. 12.

Fig. 11.—Olivine Gabbro in White Light.

The mineral with high relief is olivine with dark, included magnetite. The gray mineral of less relief is augite. The white mineral is plagioclase. The black mineral is magnetite.

Fig. 12.—The Same as Fig. 11, but taken with Crossed Nicols.

The rock was thus half diorite, half gabbro in its characters, but it seemed to be only a variation from the usual diorite and not a separate intrusion.

Olivine-Gabbro.—A still more marked departure from the usual diorite is found on the Santa Rosa claim, on the southwest extension of Providencia Hill. Excellent fresh rock was gathered on the surface in an area of noticeable but moderate magnetic attraction. When the rock was examined microscopically it proved to be a typical olivine-gabbro, Figs. 11 and 12. Plagioclase, pale-green augite, pale-green olivine, and unusually coarse bits of magnetite are its components. The magnetite develops curious skeleton growths. A very little brownish-green hornblende appears in parallel growths with the augite. It may be that the gabbro is a separate and distinct mass from the ordinary diorite which contains on the north side of Providencia Hill the new Providencia mine and on the south side the Alfredo. Forest growth and concealed exposures made the close tracing of relationships difficult, if at all practicable. The rocks also are not easy to identify sharply without constant appeal to the microscope. Olivine-gabbro makes one apprehensive of titanium in any ore appearing where it constitutes the wall rock.

Wall-Rocks of the Juragua Mines.—The same diorite as has been described above from the Spanish-American company's lands appears as the chief wall rock of the Juragua company's East Mine and West Mne. In the slide from the former as shown in part, in the photomicrograph, Fig. 10, plagioclase, a little orthoclase, hornblende, a little biiotite and one small augite crystal together with magnetite and a little pyrite were observed. In the latter, in a more finely crystalline rock, were plagioclase, hornblende, a little quartz and much magnetite.

Inclusions in the Diorite.—North of Vinent along the Daiquiri River and also on the slopes of Providencia Hill, the diorite is richly provided with angular inclusions of the same dense, dark diorite which we find in the granite, as earlier described. The component minerals of the inclusions are markedly well bounded. Rectangular feldspars, prismatic, sharply six-sided hornblendes and bits of magnetite chiefly constitute them. No pronounced contrasts in mineralogy can be detected as against the usual diorite, but there is more hornblende and the texture is different from the coarser varieties. Apparently the inclusions represent some older solidified rock.

In a recent valuable paper on the Boulder Batholith of Montana, Paul Billingsley[9] has remarked the abundance of dark, angular dioritic inclusions in this famous granite. He has suggested that they represent an older and first chilled basic shell, afterward broken up and included in the later arising granite from a new propulsive effort. The contrasts of the inclusions are so great at Daiquiri, especially with the granite, and diorite is such a common rock, the writer leans to considering the fragments as derived from an older mass, broken up by a newer intrusion.

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[9] *Bulletin* No. 97, Jan., 1915, p. 42.

Alteration of Diorite near the Ore.—Near the orebodies the diorites are noticeably altered and become very unsatisfactory for microscopic study. Epidote appears in secondary veinlets and sometimes in larger veins. Next these veins of epidote the diorite becomes dense and greenish gray, passing practically into a felsite. The process generally summed up under the word propylitization has taken place on a large scale. Dark silicates yield chlorite which stains the whole mass of the rock green. Feldspars become muddy and kaolinized and finally a mass of secondary products must be studied in which one faintly traces the outlines and structures of once fresh crystals. So constant is this change near the orebodies that one cannot avoid the conviction that thermal waters have been actively circulating.

White Siliceous Rocks.—In all the larger mines, dikes or at least longitudinal belts of white feldspathic and sometimes quartzose rock are visible, at times in close association with the orebodies and on one side. Lee Reifsneider, the superintendent of the mines on Lola Hill, remarked of this light-colored rock that in passing from diorite into it he looked for ore on the far side; but in passing from ore into it, he anticipated barren rock beyond. In the field it was difficult to decide on the nature of these rocks. They seemed in most instances silicified diorite and were considered to be the result of some thermal action associated with the ore formation. This explanation is not impossible. A specimen was gathered in the Lola-San Antonio workings which under the microscope proved to be a shattered quartz vein with some isotropic mineral filling the cracks, Fig. 13.

A second specimen collected in the Magdalena mine is a siliceous dike of granite or quartz-diorite, akin to pegmatites. Quartz, acidic plagioclase, sometimes in micro-pegmatitic intergrowths, and a little epidote made up the slide, Fig. 14. A third specimen from the East Mine of the Juragua company is a beautiful case of a micro-pegmatitic granite dike and is illustrated in Fig. 15. A fourth case from the east side of the West Mine of the same company appears to be an old sandstone. The first three cases were believed to be in some way associated with the ore formation, since the acidic rock seemed often to form one wall of the ore. The fourth case, if correctly interpreted must be older than the ore. But besides all these instances there is a fifth on the west side of the West Mine at Firmeza, where an undoubted granitic or pegmatitic dike with an offshoot cuts, or in earlier exposures did cut, the ore. It is a richly quartzose rock with quartz in acidic plagioclase, so as often to suggest micropegmatite, Fig. 16. A little hornblende and magnetite are also discernible. In the small branch of the larger dike, biotite is the dark silicate. Of these rocks, the sandstone, if correctly interpreted, stands by itself. The first three certainly seemed to have some close relationships with the ore. They might conceivably be later, but the field relations did not suggest it. At the Concordia Mine a white, feldspathic rock was also found associated with the ore. Under the microscope it is chiefly a finely crystalline feldspathic aggregate, reminding one of nothing so much as somewhat badly altered trachytes or bostonite dikes. Blotches of less definite igneous textures appear as well and make the rock very puzzling for sharp determination. Probably the original diorite has been severely affected by thermal

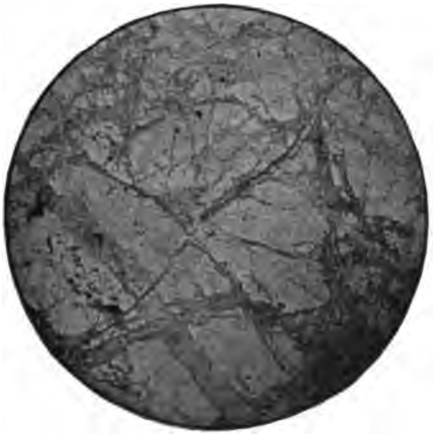

Fig. 13.

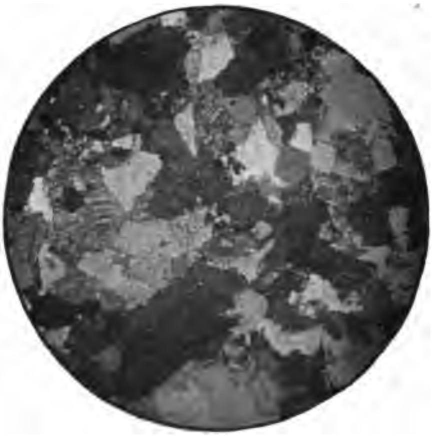

Fig. 14.

Fig. 13.—Shattered Quartz Vein, Lola Mine.

Fig. 14.—Very Acidic Quartz Diorite, consisting of Quartz, the most abundant Mineral in Clear, White, Light or Dark Gray Areas; Micropegmatite, the Mottled Areas; and Epidote, the Dark Shreds. Magdalena Mine.

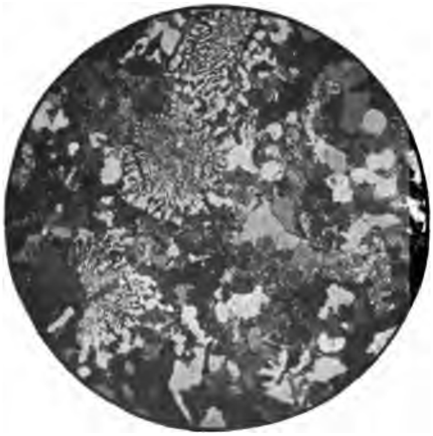

Fig. 15.

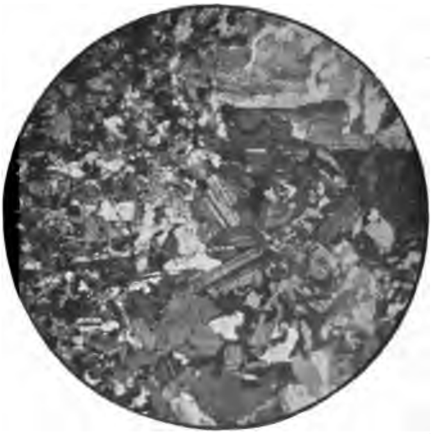

Fig. 16.

Fig. 15.—Micro-pegmatite from a Dike in the East Mine, Firmeza.

Fig. 16.—Acidic Dike, West Mine, Firmeza.
Acid plagioclase with micro-pegmatitic quartz on the right. Fine-grained diorite, forming the wall rock on the left.

waters, has lost its dark silicates and has changed over to this feldspathic residue.

In the New El Norte mine of the Daiquiri group a light-colored porphyritic dike with branches penetrates the ore in Cut 3 as illustrated in Fig. 17. It is clearly later than the ore and proves under the microscope to be a dacite porphyry. Phenocrysts of plagioclase, rarer orthoclase, and quartz are set in a finely crystalline groundmass of quartz and

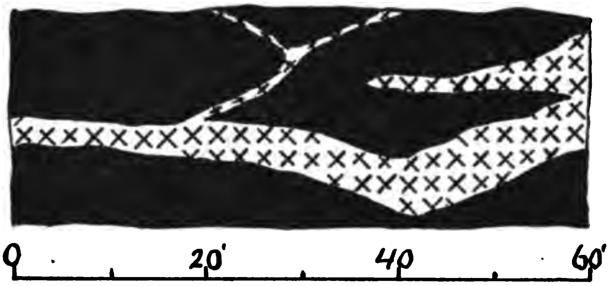

FIG. 17.—RAMIFYING DIKE OR THIN SILL OF DACITE-PORPHYRY CUTTING THE ORE ON CUT 3, NEW NORTE MINE.

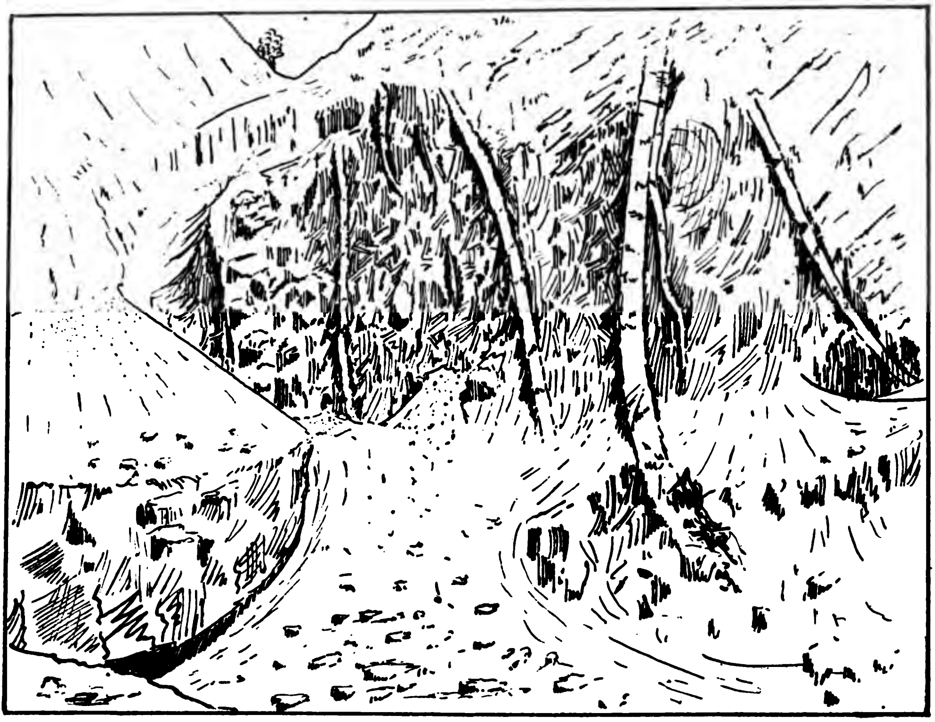

FIG. 18.—NARROW DIORITIC DIKES, CUTTING ORE IN WEST MINE, FIRMEZA. Sketched from a Photograph taken by W. L. Cummings.

feldspar with microgranitic texture. Tracts of biotite may be determined in the surviving chlorite.

Diorite Dikes.—In the larger mines, such as those on Lola Hill, the main supply of the Spanish-American company, and in the East and West Mines of the Juragua company, as well as in some of the smaller openings, narrow black dikes cut wall rocks and ore alike. They are undoubtedly later than both and have entered long after the deposition of the ore. Since the best and freshest ones collected cut also the light-colored feldspathic rock at the West Mine of the Juragua company, the narrow dikes are believed to be later than the white rocks. There seem to be two sets of dark dikes. In Stope 6 of the underground work in the Magdalena mine a later one crosses an earlier one. Six from the Lola Hill have been examined in thin section.

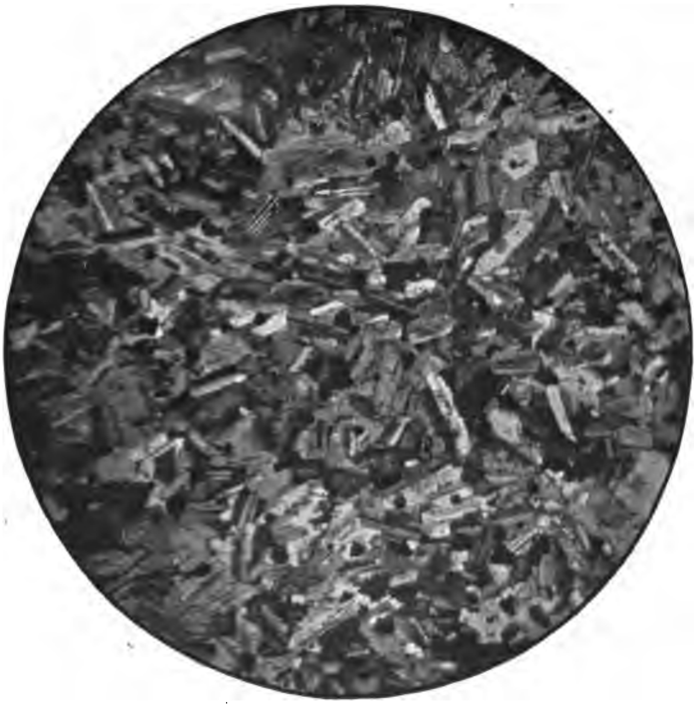

FIG. 19.—FINE-GRAINED DIORITE FROM A NARROW DIKE, WEST MINE, FIRMEZA, AND SHOWING CHARACTERISTIC TEXTURE.
The rods and squares are plagioclase, which constitutes the larger part of the rock. The dark minerals are hornblende and magnetite.

One variety is a very feldspathic diorite porphyry which cuts the ore of the Magdalena mine. Phenocrysts of zonal plagioclase are set in a groundmass of smaller plagioclases, all pretty sharply bounded by their own faces. The phenocrysts reach 2.5 mm. but the groundmass is made up of components from a quarter to a tenth this size. A few six-sided chlorite pseudomorphs after hornblende prisms remain. A little magnetite is dotted through and there is some secondary calcite.

The narrower dikes from Lola Hill have much smaller components of which plagioclase is the chief. The plagioclase shows a marked tendency to develop rectangular or square cross-sections. It is a basic variety. The dark silicates are more abundant than in the diorite prophyry just mentioned. Presumably the common one was hornblende but only chlorite survives to indicate it. The dikes have much fine magnetite. Calcite and epidote are also often in the slides.

An unusually fresh dike from the West Mine, Juragua, was 1 ft. thick. It contained, as shown in Fig. 19, beautifully fresh plagioclase which affords rectangular sections. Irregular bits and shreds of leek-green hornblende in less amount than the plagioclases are scattered through the slide, as are fine bits of magnetite. The slide reminds one strongly of diabases, with their lath-shaped plagioclases and irregular augites, but there seems no good reason to regard the hornblende as secondary after augite.

These dark, basic dikes undoubtedly represent the last igneous rocks to enter. They are closely akin to the chief wall rocks of the ore. They are sometimes associated with copper minerals along their borders, so that green copper stains now appear near them and in mining the iron ore it has been possible to throw to one side a few tons, rich enough in copper to send to El Cobre near Santiago. Usually, however, only a few green stains are the result.

SUMMARY OF ROCKS ASSOCIATED WITH THE ORE

The general associates of the ore are now before us. They constitute a complex and in some respects a very puzzling group to arrange in a logical series and to place in their time relations. One cannot help associating the granitic or pegmatitic dikes with some large parent body. The natural one is the intrusive granite mentioned at the outset. Yet this granite has produced contact zones on the older limestone with orebodies, whereas the granitic and quartz-porphyry dikes are, in two cases at least, later than the large orebodies. We can only suspect the possible connection without being able to prove it. If true, it would move the Playa granite into a place later in time than the main ore deposition or at least place its expiring effects and offshoots at this late period.

The great mass of diorite certainly preceded the main ore formation, unless one concludes with Dr. Spencer that the masses of ore have been torn off from older formations below and floated upward to their present positions. Details of the ore now to be given cast much doubt upon this hypothesis. Finally many basic dikes and some acidic ones followed the ore formation.

THE ORE

The ore is mainly a fine-grained magnetite. Dr. Spencer noted this character although almost all the earlier descriptions speak of it as specular hematite. A belief prevails at the mines that it is a fine aggregate of both specular hematite and magnetite, and that analysis has shown this to be the case. Some light might be thrown upon the question by determining the ferrous iron and its relations to the ferric, but care would of necessity

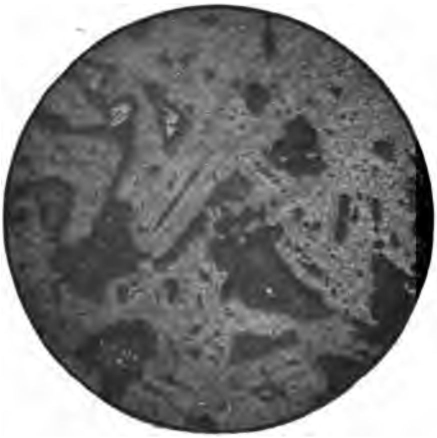

FIG. 20.

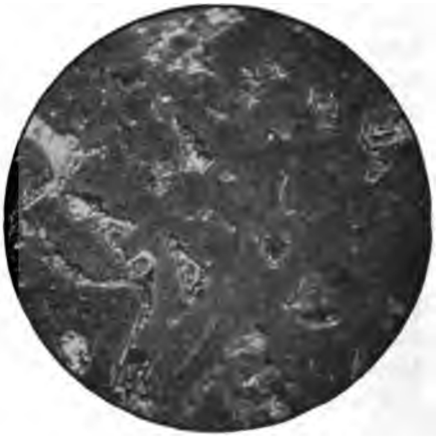

FIG. 21.

FIG. 20.—POLISHED SLAB OF ORE FROM LOLA HILL, BY DIRECTLY REFLECTED LIGHT. The light-gray areas, showing the structure of lath-shaped crystals of plagioclase, are magnetite. The irregular dark patches are quartz and garnet.

FIG. 21.—NEARLY THE SAME VIEW AS FIG. 20, BUT WITH OBLIQUE ILLUMINATION SO AS TO BRING OUT THE QUARTZ IN THE DEPRESSIONS.

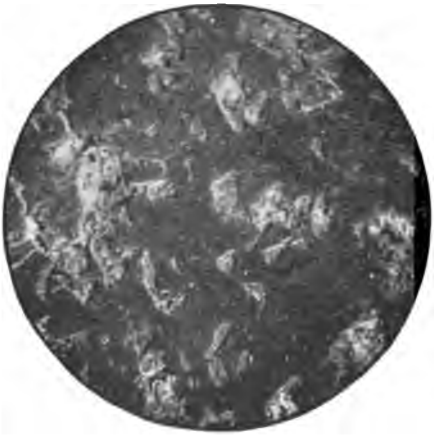

FIG. 22.

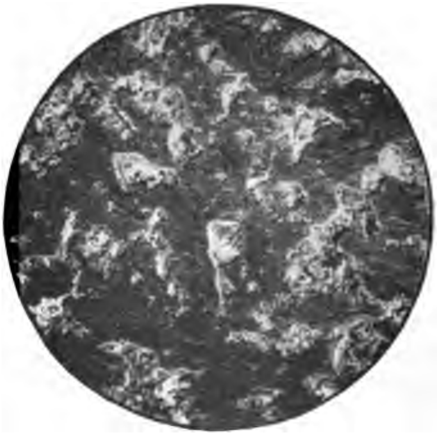

FIG. 23.

FIG. 22.—ANOTHER AREA WITH PYRITE STANDING OUT IN RELIEF FROM THE MAGNETITE.

FIG. 23.—A STILL DIFFERENT AREA WITH PYRITE IN RELIEF.

be exercised against pyrite which is widespread in small amounts. A representative suite of specimens taken at intervals from one end of the Lola Hill mines to the other has been tried with the magnet. All but one or two were immediately attracted by it. One or two were inert. We are forced to conclude that the greater part of the ore is magnetite. In the contact zones, as represented in the smaller mines, rosettes of specular ore are not infrequently to be seen.

In a polished slab of ore, illustrated in Figs. 20 to 23 inclusive, the structure is a cellular one with quartz, garnet and pyrite in the interstices of the magnetite. The magnetite itself looks like nothing so much as a replacement of lath-shaped crystals of feldspar such as we often see in the diorite itself. One can hardly resist the impression that in the slab some original rock texture is preserved. This texture is most easily explained as due to a replacement of the diorite with ore.

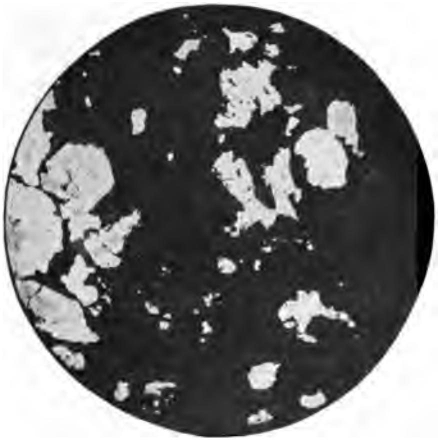

Fig. 24.

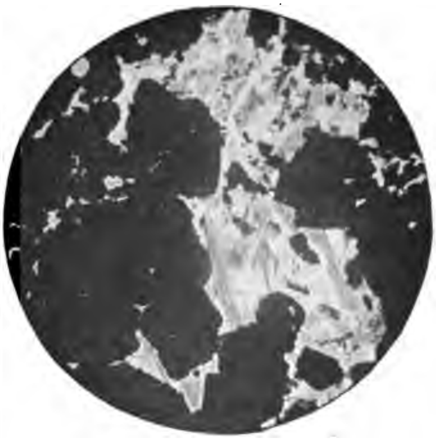

Fig. 25.

Fig. 24.—Quartz Intergrown with Magnetite. Magdalena Mine.
Fig. 25.—Quartz Intergrown with Magnetite in Ore from Firmeza.

The ore from the portions of the mines above the water level is open textured and cellular because of the oxidation and removal of pyrite, which in the deeper workings appears in unaltered condition. On Lola Hill in the northern workings in the Madgalena claim one notes little in the way of admixture in the ore except pyrite, which increases with depth. But as we pass to the south and into the Lola and San Antonio workings we find notable and significant amounts of quartz. The quartz does not seem to be a secondary and late introduction into cavities in older magnetite, but an original component of the ore. Blades, minute, irregular, angular lumps and dust of magnetite are contained in the quartz. Some magnetite at least apparently crystallized with it (Fig. 24). Similar relations obtain at Firmeza, as shown in Fig. 25.

To a small extent in the ore of the Magdalena mine, but to an increasing degree as one goes south through the Lola into the San Antonio, garnet becomes a component in the ore. Some specimens are a mere mass of garnet and magnetite in blotchy intergrowth. The garnet may even be the chief mineral. Garnet and quartz may both be intergrown and associated in the ore, as shown in Fig. 26. Garnet and calcite are also associated in the ore, Fig. 28. The relations of the garnet and magnetite are so intimate that they must have been produced by the same general process. In one slide from a specimen of ore from the middle of the orebody on a section in Lola Hill, 1,050 ft. from the north end of the Magdalena, a brightly refracting, rather high-index, optically positive mineral was found in a slide, which is otherwise chiefly garnet and magnetite. The mineral is irregularly cracked but has also a rectangular cleavage and parallel extinction. It appears to be olivine, although the association is unusual for olivine (Fig. 27). Still one is reminded of the olivine found by Charles Palache and described by C. H. Warren[10] in veins in the Cumberland, R. I., titaniferous magnetite, and obviously an after-effect in an igneous mass. The magnetite at times shows much epidote. The epidote does not seem to be necessarily secondary but to belong with the garnet as one of the original crystallizations in the process of ore deposition.

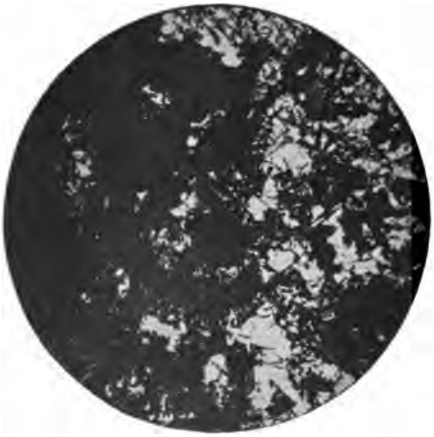

FIG. 26.

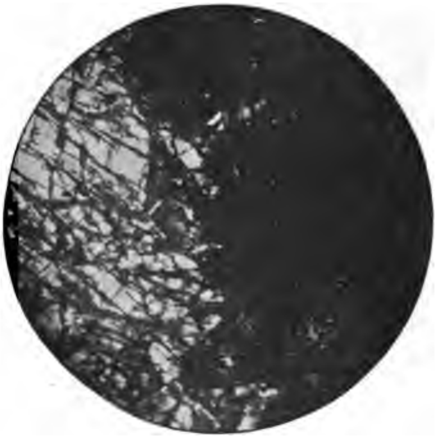

FIG. 27.

FIG. 26.—QUARTZ AND GARNETS INTERGROWN WITH MAGNETITE. MAGDALENA MINE.

FIG. 27.—OLIVINE INTERGROWN WITH MAGNETITE. LOLA MINE.

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[10] C. H. Warren. Petrography and Mineralogy of Iron Mine Hill, Cumberland, R. I. *American Journal of Science*, 4th ser., vol. xxv, pp. 35, 36 (1908). An earlier case described from the Alps by E. Weinschenk is cited by Dr. Warren. Weinschenk attributes to it a method of origin such as is here supported for the Daiquiri case.

A sample of garnet was collected in the San Antonio mine and was kindly analyzed by R. B. Herr, Jr., chemist of the Spanish-American Iron Co.

		Molecular Ratio			Molecular Ratio
SiO_2	35.23	587	CaO	30.22	537
Al_2O_3	0.82	8	S	0.01	...
Fe_2O_3	30.11	198	P_2O_5	0.05	...
FeO	0.71	10	H_2O +	0.15	...
MnO	0.10	1	$CaCO_3$	2.56	...
MgO	trace	...			
			Total	99.85	

When recast for molecular ratios we have: SiO_2, 587; Fe_2O_3 (198) + Al_2O_3 (8), 206; CaO (537) + FeO (10) + MnO (1), 548. The sesquibases are thus too high, or the silica somewhat low and the protoxides very much too low for an exact garnet formula of 3 SiO_2, $(FeAl)_2O_3$, 3(Ca, Fe, Mn)O, but it is clear that the garnet is predominantly the lime-iron garnet, andradite, most characteristically developed in the contact zones. There was probably some magnetite or hematite in the sample as 2.5 per cent. Fe_2O_3 was soluble in dilute hydrochloric acid.

Along the northeastern side of the San Antonio workings, in the summer of 1914, was an extended streak of garnet rock. Excellent crystallized specimens of a brownish-red variety could at times be obtained where they had projected into a cavity which had later been filled with calcite. This streak or belt of garnet in the diorite seemed attributable to the same general processes as the ore, but it may have been formed from a long included slab or series of slabs of limestone.

Very similar garnet rock may at times be observed in the mines of the Juragua company but the rocks have not been studied in such detail as those at Daiquiri. From the East mine a specimen was gathered which reveals under the microscope pale-green garnet with the ore and quartz mingled. Apparently the quartz was the last mineral to be deposited. In the slide amphibole is sparingly present. From the Concordia mine, garnet rock was gathered which is likewise pale green under the microscope. The garnets are possessed of optical anomalies.

With the ore of some of the smaller mines of the Spanish-American company is associated a black, heavy rock called by the Spanish miners "malo mineral" or "bad ore," and sometimes because of its blue-black color "piedra azul" or blue rock. Specimens were collected both from the New Norte and from the Old Norte mines, and similar occurrences do not fail in other openings. The writer was not always able to distinguish the piedra azul from the ore itself, but on request the miners invariably decided without hesitation. The piedra azul is shown by the microscope to be a phase of the mineralization which led to the ore. The commonest variety is a very fine-grained aggregate of magnetite and augite, Fig. 29. One would at first take it to be a narrow dike, but in another piece a fine-grained mixture of garnet and pyroxene made up the rock, allying it with the contact effects. The best explanation is therefore that while the mineralizing agents in one place produced ore by replacement of the diorite with magnetite, they may have afforded dense and closely crystalline bodies of magnetite and augite or both these and garnet, close alongside.

One other striking feature of the larger mines of the Spanish-American company should be emphasized before passing to the smaller openings which are based on characteristic contact deposits between intrusives and limestones. One cannot study the great open cuts in Lola Hill without being impressed with the evidences of movement and crushing. The diorite is crushed along planes which run in close parallelism with the ore. It is squeezed and sheared around the buttress-like masses of magnetite which have presented a resisting front to the movement, but which probably antedate some though not all of it. The alteration of the diorite is excessive and it has apparently been subject to some influence, especially in the vicinity of the ore which has almost obliterated the hornblende and feldspar in favor of chloritic greenstone. In this crushed zone, traversing Lola Hill like a backbone, the great slab-like masses of ore are distributed along a northwest and southeast line.

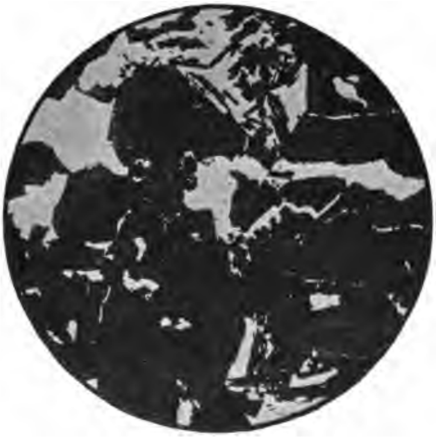

Fig. 28.

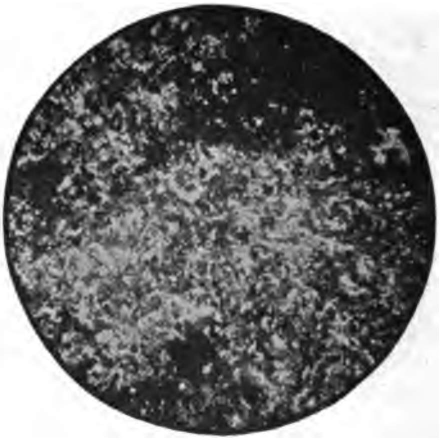

Fig. 29.

Fig. 28.—Calcite in Magnetite from one of the Mines on Lola Hill.

Fig. 29.—Piedra Azul or Malo Mineral, a Fine-Grained Intergrowth of Magnetite and Augite.

Dr. Spencer sketched several cross-sections of the orebodies when at the mines in 1901, which are reproduced in Figs. 3 to 6, along with one from Firmeza. To those in Figs. 3 to 6, is added one more, Fig. 30, reproduced from panoramic photographs of Lola Hill made by the photog-

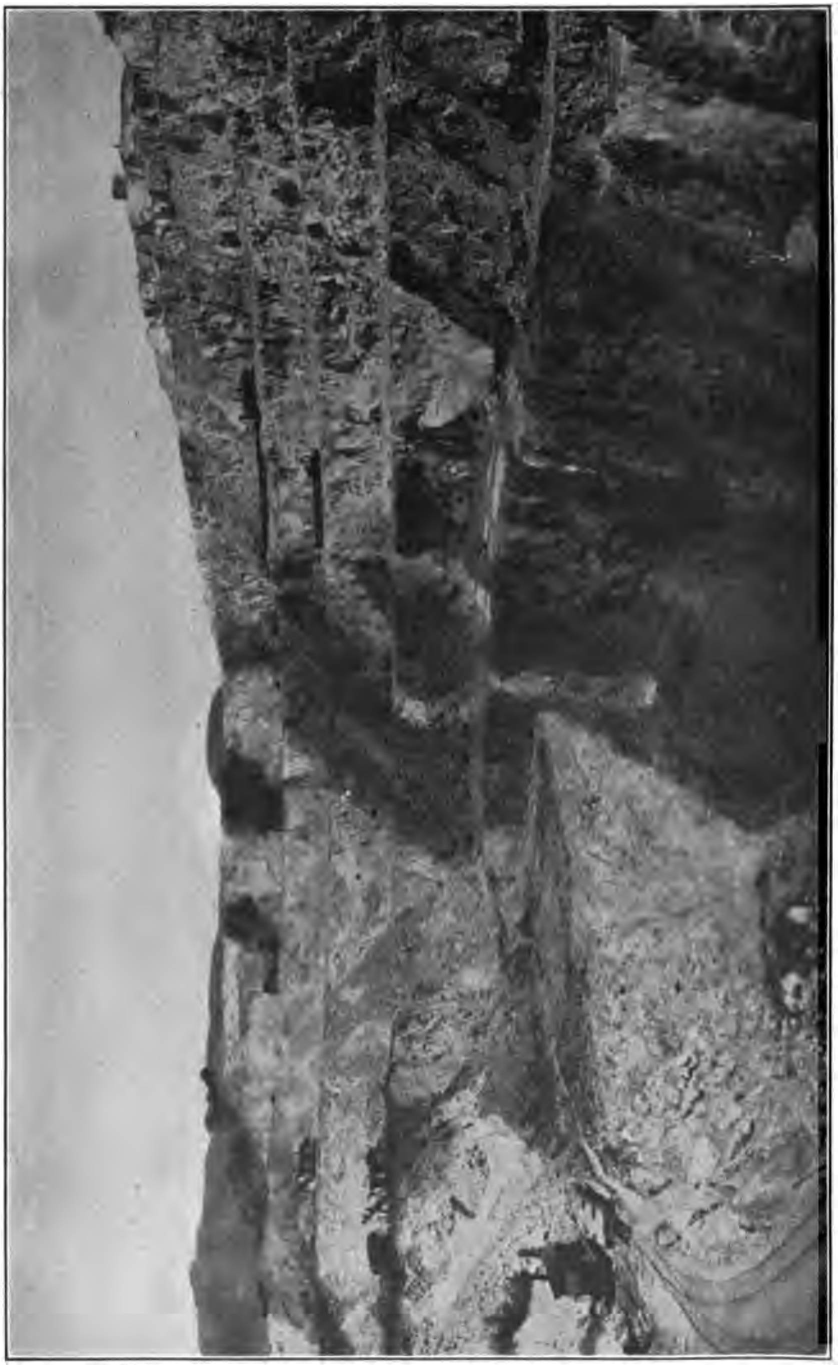

FIG. 30.—PART OF A PANORAMIC VIEW OF LOLA HILL, REPRODUCED TO ILLUSTRATE THE HUGE, TABULAR MASSES OF ORE.

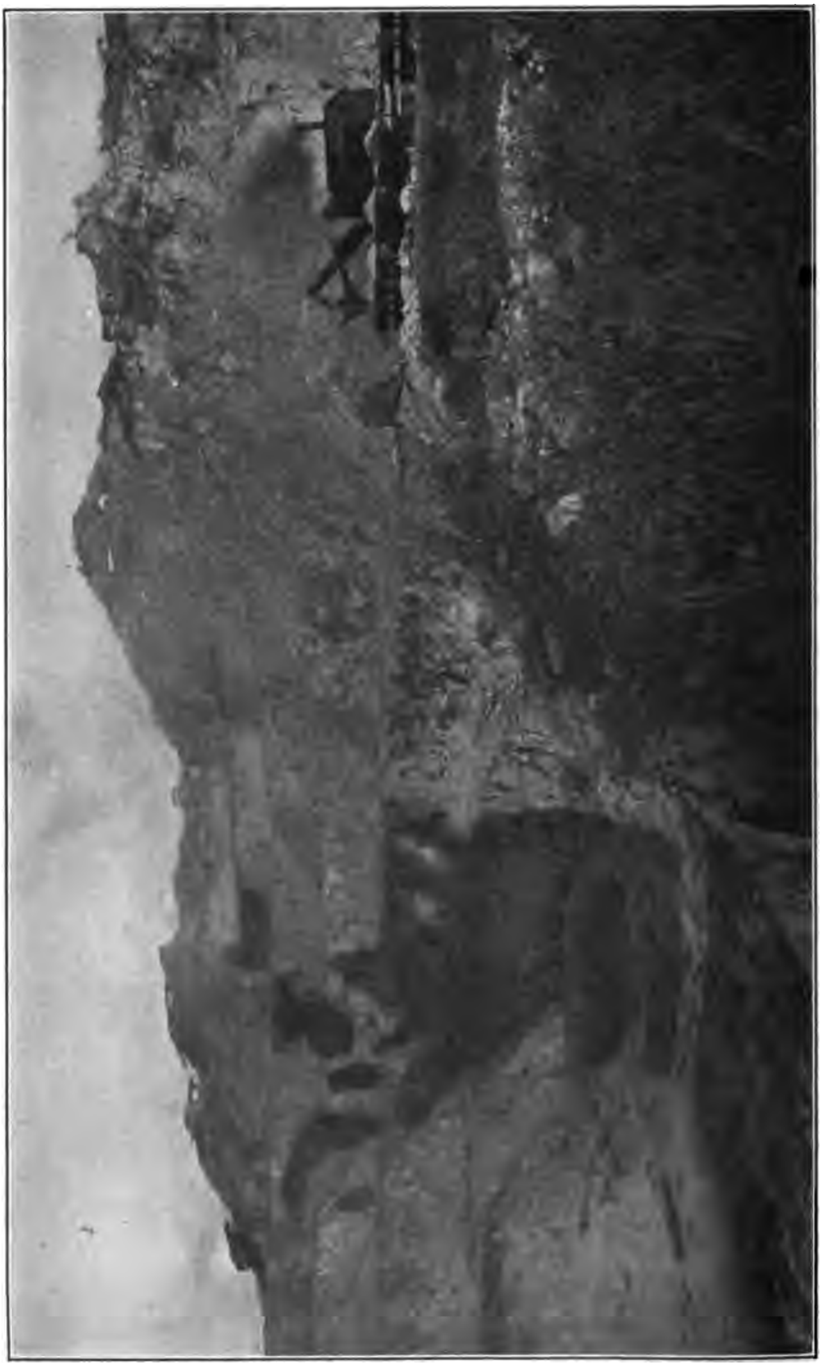

FIG. 30a.—A SECOND PANORAMA OF LOLA HILL TAKEN AT A DIFFERENT STAGE IN THE OPERATIONS AND SHOWING A DIFFERENT DISTRIBUTION OF OREBODIES FROM FIG. 30.

rapher of the Spanish-American company, Francisco Crego. The sketches only afford cross-sections, but we can justifiably imagine that they continue for relatively long distances at right angles to these cross-sections. They are much deeper than wide and in the aggregate, so far as mining has shown, they are on Lola Hill longer than deep. The ore has been followed for 2,800 ft. from the north extreme of the Magdalena mine to the south extreme of the San Antonio. The old summit of the hill with outcropping ore was about 1,000 ft. above tide, while the lowest exploratory tunnel which was cut is 500 or 600 ft. lower.

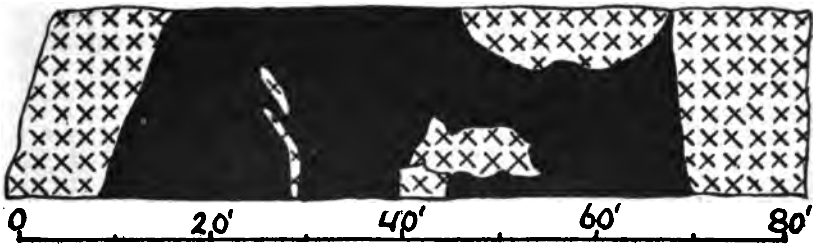

FIG. 31.—SKETCH OF CUT 4, NEW NORTE, TO ILLUSTRATE SURVIVING FRAGMENTS OF DIORITE IN ORE.

Thus the whole arrangement of the orebodies is an elongated one, such as would arise from mineralization along a fissured strip. One can hardly imagine such a long and relatively narrow slab or series of slabs to be torn off in depth and floated up edgewise into their present position. On the contrary, the structure of the ore, reproducing the interlacing feldspar, the associated pyrite, the quartz and the garnets constantly remind us of replacements by just such emissions as we generally believe have produced the contact metamorphic zones.

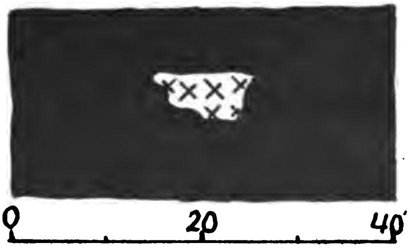

FIG. 32.—MASS OF DIORITE IN ORE, CUT 3, NEW NORTE.
The diorite appeared to be a surviving fragment from a fault breccia.

Another very striking feature is this: When we prolong the general strike of the orebodies on Lola Hill to the southeast across the valley of the Daiquiri River, we run almost, if not quite exactly, into the Providencia mine on Providencia Hill. Along the same general line over the top and down the other side of the hill is the Alfredo, prolonged again beyond another valley by the Norte and New Norte. They all line up as well as one could expect along a great belt of crushing and faulting. They display much the same characters from end to end, but the greatest mineralization is on Lola Hill.

Effects of crushing do not fail in the smaller mines. Fig. 31 is reproduced from a sketch made in Cut 4 of the New Norte. Fragments of wall rock remained in the mass of ore. In another part of the hill in Cut 3 of the New Norte, relations shown in Fig. 32 were found. A piece of wall rock had survived in the ore along a zone of crushing.

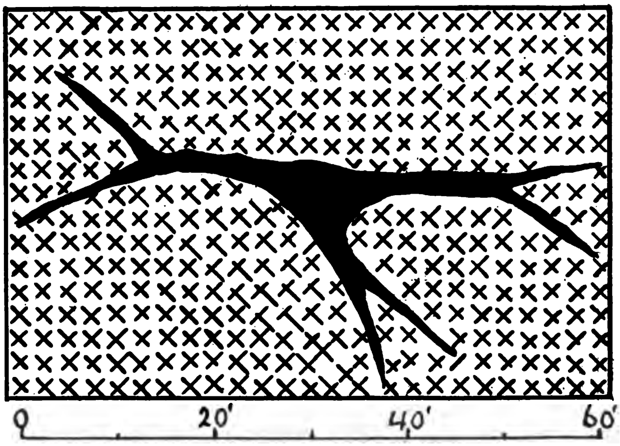

FIG. 33.—RAMIFYING VEIN OF MAGNETITE WHICH APPARENTLY HAD REPLACED THE DIORITE ALONG INTERSECTING FISSURES. ABANDONED WORKINGS OF THE ALFREDO MINE.

One is led to the conclusion that while the diorite mass was still hot in the depths or after it had consolidated and had been penetrated by some other and still hot intrusive in depth, a pronounced northwest and southeast fissured zone was formed, up through which came the emissions, fluid or gaseous, which brought the iron for the ore, the pyrite, the garnet and the epidote; the sulphur for the pyrite; and the silica for the quartz, the garnet and the epidote. The lime required by the garnet and the epidote may have been derived from the plagioclase and hornblende of the diorite, or from included blocks of limestone, or deep-lying limestone, or from several of these sources. The iron probably came up as chloride as has been favored by several writers on contact zones.[11] The relations at Lola Hill are on a large scale not very different from the dike-like masses of magnetite in the intrusive rock of the Iron Springs

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[11] C. K. Leith and E. C. Harder. The Iron Ores of the Iron Springs District, Southern Utah, *Bulletin No.* 338, *U. S. Geological Survey*, p. 77 (1908). V. Goldschmidt. Contact-zones in the Christiania District, Norway.

District of Utah, but the largest masses are in contact zones reversing the Daiquiri relationships. E. P. Jennings[12] has considered the dike-like masses at the Utah locality to be intrusive igneous rock. The same conception has been very seriously considered by some observers for Daiquiri although it has not been recorded in print. Nevertheless, the writer favors replacement along a fissured belt as having the strongest claims to confidence (Fig. 33). In drawing conclusions everyone should keep before him all the points made regarding the shape, associations, lineal distribution, microscopic texture and mineralogy of the orebodies.

The Distinctive Contact Zones

There are a number of smaller mines now largely abandoned which are obvious contact zones. They are all in close association with limestone and igneous rock and bear all the marks of having originated by the action of the latter on the former. The two old workings in the Concepcion and San Sebastian are particularly good examples. The former has two pits, a lower on the north side of the hill and not much above the railroad, and an upper and smaller one on top of the hill farther south. The lower cut is entered through granite. A broad zone of epidote succeeds in which were streaks and irregular bodies of ore. The granite had obviously produced from the limestone a contact zone with epidote as almost the only lime-silicate. Together with epidote, bodies of magnetite also resulted. When studied under the microscope the iron ore is also intimately intergrown with garnet and quartz as shown in Fig. 34. On top of the hill the lime-silicate changes to garnet. No obvious reason appeared for the change, but as the two minerals are so closely akin no great difference of physical conditions was probably involved. The Sebastian pit is said to have the ore in garnet also, but darkness prevented the writer from seeing it, on the trip to these claims. On the east side of the Sebastian claim, however, along the Berraco railroad, one passes through cuts of granite when riding south, and then finds a large knob of coarsely crystalline, white marble, extensively changed in places to coarse, prismatic epidote which is illustrated in Fig. 35. The relations are much the same as at the Concepcion lower pit, but no magnetite was noted. No one at all familiar with contact zones produced by intrusive rocks from limestones would hesitate for a moment over the interpretation of these exposures.

The Barcelona claim with two workings of modest size is a case both of ore in the diorite along an apparent vein and in a small contact zone visibly lying between the limestone and the diorite. One walks along

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[12] E. P. Jennings. Origin of the Magnetite Iron-Ores of Iron County, Utah, *Trans.*, xxxv, 338 (1905).

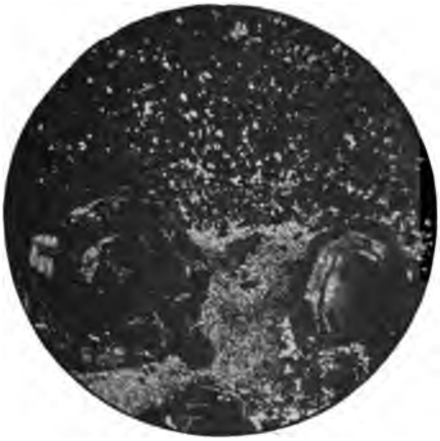

FIG. 34.

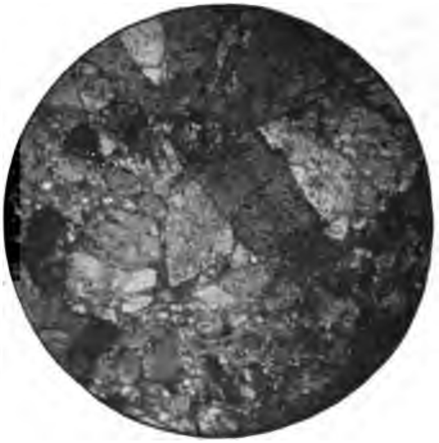

FIG. 35.

FIG. 34.—RELATIVELY LARGE GARNETS, SHOWING DOUBLE REFRACTION AND IN A FINE-GRAINED INTERGROWTH OF QUARTZ AND MAGNETITE. CONCEPCION MINE.
FIG. 35.—EPIDOTE ROCK, FROM A CONTACT ZONE ON THE BERRACO R.R., PRODUCED BY GRANITE FROM LIMESTONE.

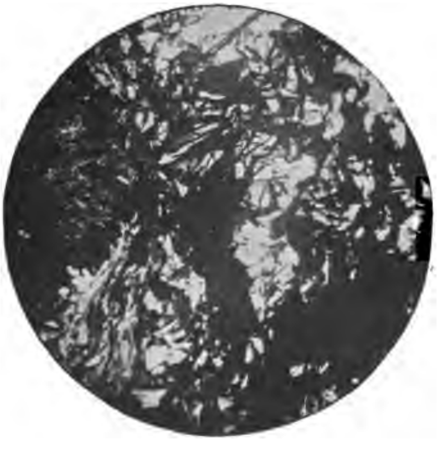

FIG. 36.

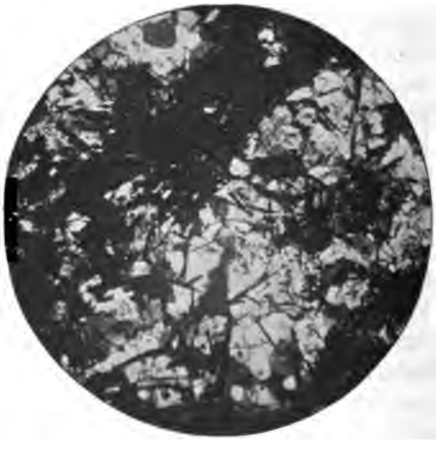

FIG. 37.

FIG. 36.—PLATY GROWTHS OF SPECULAR HEMATITE IN QUARTZ. FAUSTO PRIMERO MINE.
FIG. 37.—THIN PLATES OF SPECULAR HEMATITE, BRANCHING FROM LARGER MASSES OF ORE, AND CONTAINED IN QUARTZ. ESCONDIDO MINE.

an exposure of limestone for 200 or 300 ft. before reaching the small contact zone. No satisfactory dip or strike could be taken, the rock was so broken and disturbed. The limestone was partly white and partly blue being apparently original blue changed in places to white by the neighboring intrusive.

The Fausto Primero claim on the southeastern portion of the tract is another contact zone. The ore is largely specular hematite in stringers and larger masses in actual limestone. The observer is forced to conclude that the iron-bearing solutions have come in through all manner of crevices and have deposited the specular hematite in irregular networks throughout the rock. Much quartz is mingled with the specularite, which forms platy intergrowths with it, as shown in Fig. 36. In other specimens gathered as a mass of green silicates the microscope revealed quartz and some minute acicular mineral, believed from its small extinction to be tremolite.

Again on the Escondido claim in a small pit, lean ore and a heavy rock, the "piedra azul," were associated. The ore in thin section revealed innumerable plates of specularite set in a quartz mosaic as shown in Fig. 37.

On the Elvira claim a large open cut has been driven so as to win the stringers and irregular masses of ore which are set in garnet and in a fibrous amphibole which is practically coarse asbestos. Asbestos as an associate of the ore was found in abundance on the Rafael claim. The Elvira rock was examined in thin sections and revealed yellow garnet, calcite and the asbestos in very fine radiating fibers, Fig. 38.

The most extensive and the most interesting of the contact zones and attendant iron-ore deposits are at Sigua, high up on a steep and picturesque mountain side looking away across a valley north to the Sierra Maestra where there were coffee plantations still being cultivated on the slopes. All stages of changes can be seen from limestone through the characteristic silicates to ore, but beyond several dikes which cross the zones and are later, no good exposures of igneous rocks were noted. The dikes are a very porphyritic rock with abundant phenocrysts of feldspar, but are too badly decomposed for microscopic study. There are also green chlorite rocks which may have once been more basic intrusives. The contact effects begin with stringers and limited masses of silicates in white limestone. Apparently as at Fausto Primero the ore solutions circulated through all the crevices and open spaces and produced the streaks of minerals by reactions with the limestone. Besides the iron ore, which is largely specularite, well crystallized garnets were produced in the calcite as is shown in Figs. 39 and 40. As the effects grow stronger and the conditions of the largest pit are reached, the silicates become more extensively developed. They may take on finely parallel growths with some surviving limestone and resemble coralline

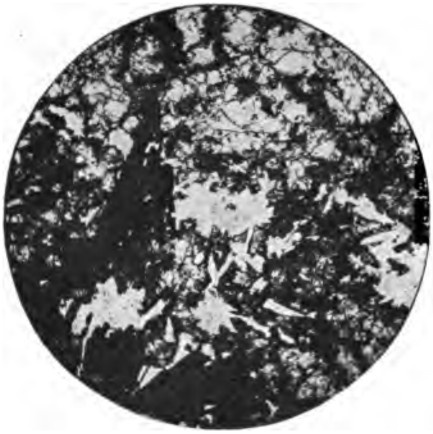

Fig. 38.

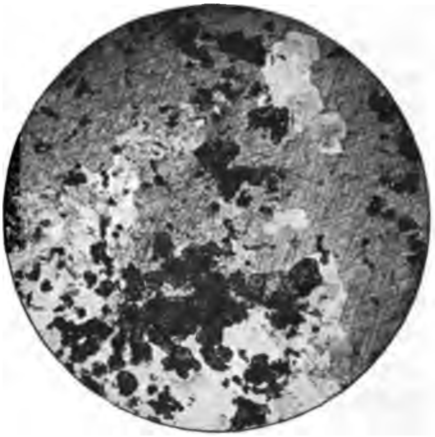

Fig. 39.

Fig. 38.—Intergrowths of Magnetite and Perhaps Specular Hematite with Finely Prismatic Amphibole and Calcite. Elvira Mine.

Fig. 39.—Relatively few Garnets (the Dark Minerals) with Quartz (the Clear White Minerals) in Calcite. Sigua Mines.

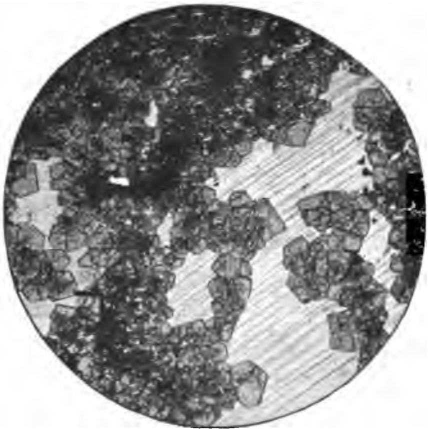

Fig. 40.

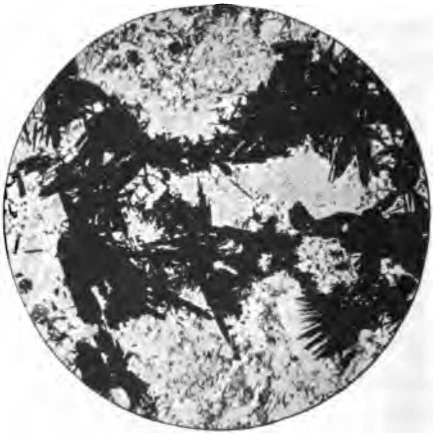

Fig. 41.

Fig. 40.—Rhombic Dodecahedrons of Garnet in Calcite. Sigua Mines.

Fig. 41.—Plates of Specular Hematite in Calcite. Sigua Mines.

structures, strongly reminiscent of Eozoon Canadense. Undoubtedly specimens like these, observed by Dr. J. P. Kimball, in the early days at Firmeza confirmed him in the idea of the replacement of coralline limestone. Sections of the contact silicates reveal all manner of interesting things. We may see yellow garnet, specularite plates, quartz, and diopside all in the same slide. Epidote appears in other slides. The specular hematite in the silicates shows a marked tendency to develop rosettes of radiating plates, Figs. 41 and 42, like roses, as one sometimes sees also in ilmenite. In the slides the plates ramify through calcite more often than through quartz. No reasonable explanation could be given of all these phenomena other than the standard one of contact effects. Some intrusive mass has fed into these limestones gases or solutions, or both, which have brought silica, iron compounds, perhaps as chlorides or oxides, and have thereby reorganized the original limestones and produced the lime-iron silicates, the quartz, and the ore. From the mining standpoint the results have led, as in most of the actual contacts in southeastern Cuba, to irregular and seldom large bodies of ore in the midst of barren silicates so that a great quantity of waste must be broken and handled to secure a commercial output. Yet the residual blocks, left on the surface by weathering, originally gave the impression of great bodies underground, and led to the construction of the Sigua standard-gauge railroad to the sea, and to the building of a pier. Floods and the insurrection wrecked the installation after 20,000 to 30,000 tons had been shipped, and since then the Spanish-American company has obtained some 60,000 tons additional by means of a narrow-

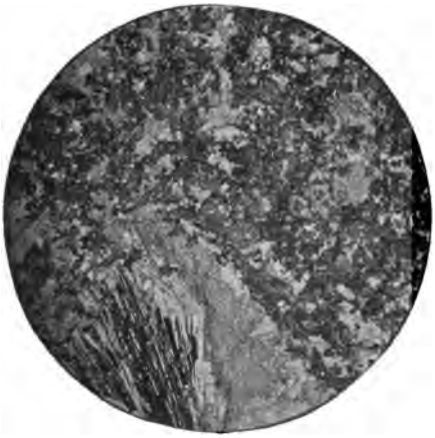

Fig. 42.

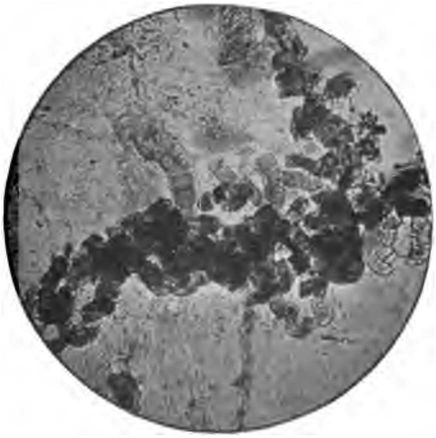

Fig 43.

Fig. 42.—Thin, Radiating Plates of Specular Hematite in Calcite, below; Finely Granular Epidote in Calcite above. Sigua Mines.

Fig. 43.—Vermicular Growths of Some Undetermined Mineral in Quartz. Actual field, 0.06 in. or 1.5 mm. Sigua Mines.

gauge extension of the Berraco railroad. Even then 50 per cent. Fe and 12 per cent. SiO_2 limits were maintained. Operations have now been abandoned.

The ore appeared in various small open cuts down in the valley below as well as on the mountain side. There is obviously some complexity to the geology, because in the gulch below the main workings a stratified rock, believed to be an old volcanic tuff and breccia, was observed. In one exposure it had a strike N. 70 W. magnetic, dip 32 N., and again N. 80 W., 20 N. It indicated a marked change in formations from the limestone. In one of the lower pits the black shaly rock was observed, which was earlier mentioned as a probable fine tuff but now altered to a green chloritic mass. On a neighboring claim, the Alfeo, float ore was being picked up off the surface but no excavations were undertaken.

Between Sigua and the other mines much volcanic or at least porphyritic rock appears on the surface. It stretches along the narrow-gauge road for some distance to the south as a green andesite. In thin section the feldspar phenocrysts are plagioclase with chloritic remains of either hornblende or augites which are no longer recognizable. The ground-mass is finely microgranitic and contains some minute untwinned mineral which may be in whole or in part orthoclase, albite or quartz, probably the two former. A little magnetite is sprinkled through the ground-mass.

To the west of Sigua a purple porphyritic rock was gathered in several places. It proves to be a trachyte. Orthoclase is much more abundant than plagioclase among the phenocrysts, but the ground-mass is very finely crystalline, almost felt-like. Alteration is extreme; only limonite remains to suggest dark silicates. The rock was gathered in the general field work and we do not know whether the trachyte appears in dikes or surface flows.

Along the Sigua extension of the Berraco narrow-gauge railroad, and about two miles in a direct line, or three miles by rail southeast of Sigua, two shafts were sunk in former years in exploration of small veins believed to contain silver. Only the dumps and the open vertical holes now remain, but small specimens of barite apparently with tetrahedrite can be picked up. The wall rock is a dense greenstone, evidently derived from some rock like the diorites. No records are available of the assay value of the ore.

RÉSUMÉ

The iron ores are the subject of chief interest and in summing up, attention will be alone directed to them. They furnish a variety of deposits ranging from those of small size in streaks in limestone associated intimately and microscopically with quartz, garnet, and epidote;

through larger and more extensive developments of the same with the practical extinction of the limestone; to the extreme of ore developed in great tabular masses in diorite, but still associated with the same quartz, garnet, and epidote, both in the large and in the microscopic way. In the large bodies pyrite appears in increasing amount as depth is gained, but it has weathered out of the ore near the surface. The smaller deposits, obviously in the contact zones, have in no case been followed deeply enough to develop much if any pyrite and we can only infer its probable presence in depth. Pyrite is quite frequently associated with the magnetites and specular hematites customarily found in contact zones. As regards the magnetite and specularite of the obvious zones, there is little question that they have been yielded by iron-bearing waters or gases from the neighboring igneous rock. So many cases have been studied and so many observers have been led to the same conclusion that little attention is demanded by this type other than the mention of the method of formation. Experience has shown orebodies in contact zones to be irregular in shape and of no uniformity of disposition in the zones. They seem not to be very large in the Daiquiri district, although in the Iron Springs District of Utah and in some other localities in the southwest they are very large.[13] In the larger and more important deposits in the Daiquiri district, such as those in Lola Hill and on Providencia Hill as developed especially in the Alfredo, and for the Old and New Norte, we cannot reasonably go far from a method of origin closely related to that of the contact zones. Their ore with its associated garnet, quartz, and epidote must be closely akin to that of the zones. The long and relatively narrow masses and the obvious location in belts of crushing and rock movement cannot be ignored. The observer is almost irresistibly led to the conclusion that emissions from some cooling, igneous mass beneath, bearing iron and silica, have come up along lines of faulting and crushing. We might justifiably wonder if where we now find the orebodies, slabs or blocks of included limestone chanced to be in the diorite and if they served as precipitants of the iron, so as at the same time to yield the associated garnets. The suggestion is by no means unreasonable for some of the smaller block-like masses, but as one appreciates the extent of the large bodies on Lola Hill, their great length and depth as compared with their width and their uniform vertical or nearly vertical position, the assumption of blocks of limestone is clearly so improbable as not to appeal to reason. The vertical and somewhat straggling line of ore which one sees at Providencia, Alfredo, and at the Old and New Norte could not be explained by it. No explanation will fit except one along the lines of vein formation with replacement of the diorite. Where the result was largely magnetite, but so mixed with silicates as to be too low in grade for ore, "piedra azul" resulted. Where the wall rocks yielded more completely to the iron solutions or vapors commercial ore resulted. The diorite was replaced with magnetite and some specular hematite, intermingled with garnet, quartz, and epidote. The lime for the admixed garnet and epidote probably come from the lime in the plagioclase or in the bisilicates of the diorite. But in the case of the streak of garnet rock in the San Antonio with admixed calcite but no appreciable iron ore, an included slab or series of slabs of limestone is not unreasonable. As stated above, veins of magnetite are known in the diorite porphyry or andesite which has developed the large contact bodies at Iron Springs Utah. E. P. Jennings has considered them iron dikes, but Leith and Harder regard them as due to emissions from the igneous mass, coming up through cracks in its outer cooled and solidified shell. In some such way as this the Cuban ores have probably been formed.

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[13] C. K. Leith and E. C. Harder. The Iron Ores of the Iron Springs District, Southern Utah, *Bulletin* 338, *U. S. Geological Survey*, p. 73 (1908). A total of 40,000,000 tons is estimated, the larger part in contact deposits, of which one reaches 15,000,000 tons.

As was inferred by F. F. Chisholm there seems no reason why the ores should be necessarily limited to relatively shallow depths, but with depth, unoxidized pyrite is quite certain to appear and to contribute relatively high sulphur. Roasting is already practiced at Firmeza.

The larger deposits thus lie along structural lines. The resistance of the ore to weathering processes has left it as the backbones and at the summits of hills whose slopes were mantled with blocks of float. Meantime the original wall rocks have weathered away and have in part been removed by erosion.

The Daiquiri ores are remarkably similar in their geological relations to a deposit near Hong Kong, China, recently described before the Institute by C. M. Weld,[14] and to others in the province of Bulacan in the Philippines as set forth by Dalburg and Pratt,[15] and associated petrographic studies by F. T. Eddingfield.[16] In each case aggregates of preponderating magnetite with specular hematite have resulted by contact metamorphism and replacement. The geological sections in Bulacan might almost be substituted for those in southeastern Cuba.

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[14] C. M. Weld. *Bulletin* No. 86, Jan., 1914, p. 177.

[15] F. A. Dalburg and Wallace E. Pratt. The Iron Ores of Bulacan Province, P. I., *Philippine Journal of Science*, vol. ix, No. 3, p. 201 (June, 1914).

[16] F. T. Eddingfield. Microscopic Study of the Bulacan Iron Ores, *Idem*, p. 263.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Tramming and Hoisting at Copper Queen Mine

BY GERALD F. G. SHERMAN,* BISBEE, ARIZ.

(San Francisco Meeting, September, 1915)

THE ore deposits of the Warren district, in which the mines of the Copper Queen Consolidated Mining Co. are situated, have been described in a number of technical publications, and will not be discussed here in detail. Certain of their characteristics, however, control methods of underground transportation and hoisting of ore.

In the Copper Queen mine, the majority of the ore has occurred in a zone encircling the west boundary of the porphyry intrusion of Sacramento Hill. It has a width varying from 800 to 1,200 ft., and a thickness of about 400 ft. It reaches the surface in the older part of the mine to the northwest, but dips to the southeast, where it is reached at 1,400 ft. below the Czar collar, in its farthest extension at present developed on Copper Queen ground. There is one major extension from the northwest end of the zone toward the west along the Czar fault, and others of minor importance.

Individual orebodies are scattered through the zone in an eccentric manner, only matched by their own irregularities of form and size. Their most general characteristics are the softness of the ore and their great horizontal rather than vertical extent. It has been estimated that the average vertical thickness of ore in the Czar and Lowell divisions is between 30 and 35 ft. It is calculated by assuming it to be uniformly distributed over its horizontally projected area.

In this zone, and for some distance above it, the ground has been subjected to intense alteration and intense but irregular oxidation. It has resulted in an enormous quantity of earthy or clayey material, which may be either ore or waste, which when wet is both heavy and tenacious. Below alteration, the ground is fairly hard, and the limestones contain primary ores differing from those heretofore considered typical of the camp, and which have not yet been thoroughly exploited.

The mine production has been drawn from many orebodies spread over a great area. A diagram map (Fig. 1), of haulage drifts and loading stations, illustrates the situation in 1914.

The effort to restrict the area of work and intensify production has not been particularly successful, for several reasons:

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*Superintendent, Mines Department, Copper Queen Consolidated Mining Co.

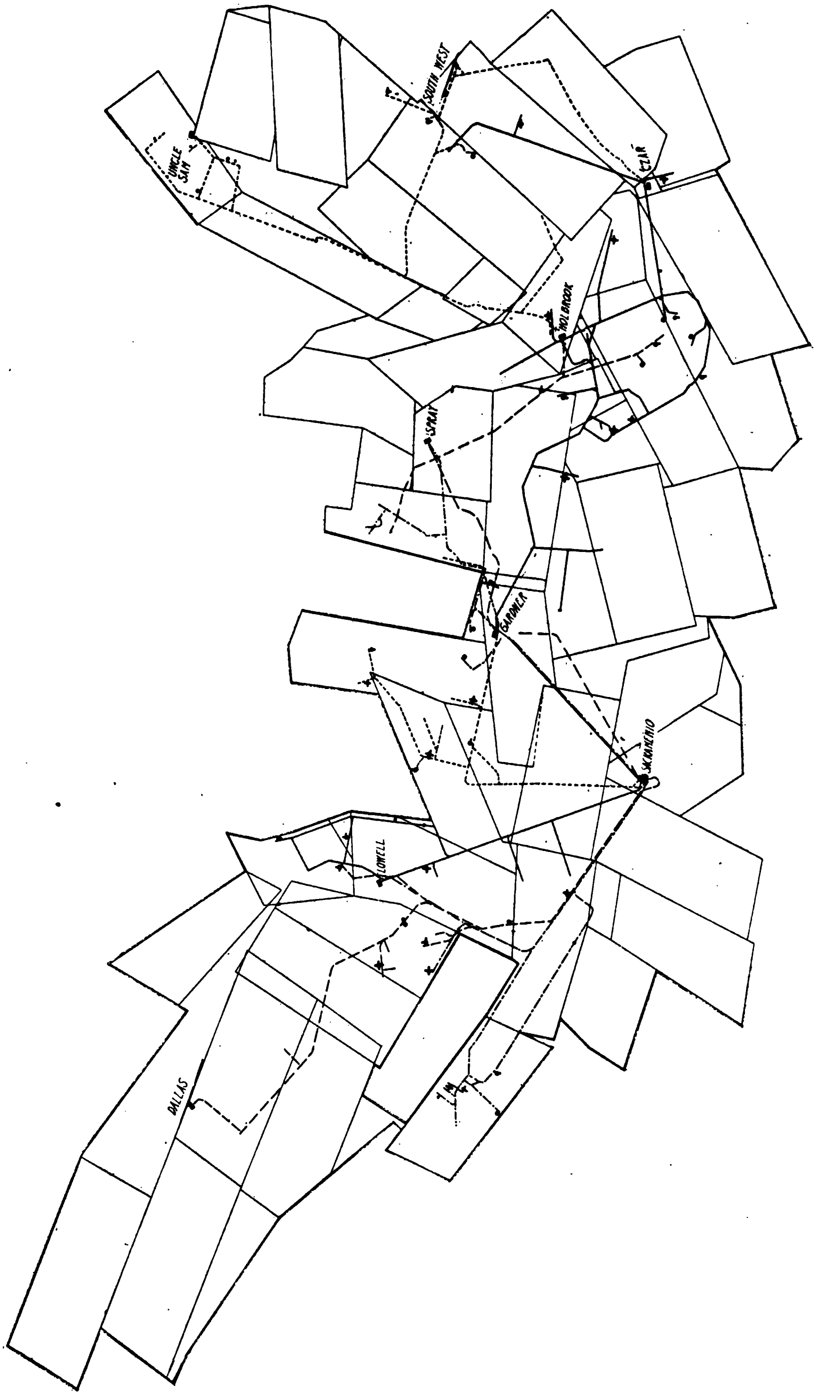

FIG. 1.—HAULAGE DRIFTS AND CHUTES IN 1914.

First: The mine has been in continuous operation for more than 30 years. During the earlier period, the costs of mining and smelting were high, and ores which are now profitable could not be worked. Exploration was not carried far enough in advance of mining to explore orebodies as a whole, and they were attacked piece-meal. Much ore was, in consequence, left in and about the old stopes. When new ground was opened to supply the demand for greater production, the old areas were not yet exhausted, their output was merely reduced. In fact, more thorough exploration and the gradually decreasing cost of mining, etc., has actually increased the production and reserves of the Czar division, the oldest part of the mine.

Second: As all ore is smelted direct, a balanced production must be made to maintain a self-fluxing mixture. Since the oldest divisions, the Czar and Holbrook, have contained highly refractory ores, their output has been limited to what the smelter could handle, and their reserves have not been exhausted as rapidly as the economies of mining alone would suggest.

Third: While many of the orebodies are large when taken as a whole, they are small in section, complicated in form and mingled with waste. They cannot be mined rapidly.

The producing area has therefore increased rather than diminished.

All the ore is not sticky or hard to handle, but some is extremely so. To one not acquainted with it, it is hard to appreciate how seriously this quality interferes with every operation of handling, and forces complications into a general plan to insure certainty of action. There is not such a large proportion of sticky ores at present as in the past, except on the 400 level, but a successful method must be prepared to take care of it wherever it appears.

To summarize briefly: The three characteristics of the deposits which affect haulage are scattered operations, heavy ground in and near the ore, and clay ores.

Old Haulage System Unsatisfactory

Prior to the introduction of the system now in use, the ore was trammed by hand to one of several shafts, where it was hoisted on cages and loaded into railroad cars for shipment to the company's smelter at Douglas. In most cases, it was dumped directly into the cars instead of loading from bins, since it was slow and hard work to get it out of storage once it was put in. An attempt was made to mix the ore when loading into the railroad cars, but without great success. Each tenth mine car was loaded into a separate car as a sample for the smelter, but the returns were not beyond question.

An unusual number of shafts were required, since active production covered a wide area. Distances to the shafts were excessive for hand tramming, and it was impossible to maintain such great lengths of drifts and track in proper condition, although heavy repairs were continuous.

The defects of the system became more and more apparent as the mine grew, and tentative plans had been discussed from time to time, but the disadvantages of any new method were more apparent than the advantages, and nothing was done.

Attention was again and more forcibly drawn to the subject when the sudden loss of the Holbrook shaft in July, 1906, diverted more ore to the Czar and Spray than they could well handle. Tramming costs were raised by the increase in the distance trammed, and by congestion at junction points or at the stations. This resulted in heavy expense, which lasted six months until the new 559-ft. shaft could be finished and equipped.

Plans were begun on a general system for tramming and hoisting, which were later extended to include a power plant, and thus began a centralization of the surface plant and work, which has only recently been completed by the construction of a central timber-framing mill and a drill- and tool-sharpening shop.

Formerly, ore was hoisted through five shafts. Beginning with the oldest and shallowest, and passing to the southeast, they were as follows:

Shaft	Depth	Production of Dry Ore per Month, 1906
Czar	417 ft. to 400 level	7,814
Holbrook	559 ft. to 500 level	17,263 (1st 6 mo. only)
Spray	1,042 ft. to 900 level	18,530
Gardner	1,032 ft. to 1,000 level	7,560
Lowell	1,215 ft. to 1,200 level	4,327
Sacramento	928 ft. to 1,000 level	....
Total for year	....	564,154

The Holbrook has since been sunk to the 600 level and the Lowell to the 1,600.

Of these, the Holbrook, Spray, and Lowell shafts were either in the ore zone or so near stoped areas that there was continual movement in them, and some little danger of a repetition of the Holbrook collapse.

Plumbings were made regularly to ascertain the extent and rapidity of movement in the new Holbrook shaft. During the first two years, the collar moved to the northeast at the rate of an inch per month. It is now 6 ft. 2½ in. N. 45° E. of the stationary part below the third level.

Although the Spray shaft is in rather solid limestone, a crack developed, which passed through the shaft and split the engine foundation.

It may be noted in passing, that the engine foundations at these shafts now consist of two massive blocks of concrete, reinforced with old rails, each heavy enough in itself to hold the engine securely, and resist the rope pull. The ground beneath was levelled off by a layer of concrete, but the surface was separated from the foundation block, so that there could be no bond which would tear the concrete block apart in case of future movement. The fissure at the Spray shaft has since widened, but with no more effect than to tilt the foundation backward, engine and all. It runs smoothly on a slope of ⅜ in. in 1 ft.

The Czar, although in rather heavy ground, was stable, and the Gardner is outside the ore zone and will probably remain true.

Each shaft was an independent unit, with its own boiler plant, etc. The equipment was miscellaneous, having been added to from time to time, and some of it was very old. The operation of the several plants was inefficient, from the number and small size of the units. The use of power was unimportant aside from that for hoisting.

New Plan Included Electric Haulage and Central Hoisting

It was proposed to equip a shaft, located in some central and safe position, for economical hoisting, where the ore might be mixed and sampled. The ore was to be drawn to the shaft by electric locomotives.

The advantages to be expected may be stated as follows:

1. The hand tram would be cut down to an average of 300 ft. or less.
2. Repairs would be reduced by diminishing the number and length of tramming drifts in bad ground. Access to stopes may be by circuitous routes to avoid heavy ground, and need not be so well maintained as is necessary when large quantities of ore must be drawn through them, the main-haulage drifts being placed beneath the ore in solid ground.
3. Fewer trammers would be required, and they could be distributed to avoid interference. There was often great congestion near the shafts, even when mining less ore than in recent years.
4. Hoisting would be cheapened by the use of skips, a more efficient engine, and a better method of transporting the ore to the railroad cars.
5. The unstable shafts would not be so essential to continuous production.
6. Shaft and cage repairs would be diminished, particularly in the moving shafts, by slowing down the hoisting speed, as could be done if only men, timber, and waste were hoisted.
7. Miscellaneous work about the shafts and auxiliary equipment might be cut down.
8. Power distribution from a central plant would be simplified and the investment for hoisting machinery reduced.
9. The ore could be thoroughly mixed and accurately sampled before shipment.

There were also certain disadvantages, aside from the cost of the improvements:

1. Haulage drifts must be driven and maintained without abandoning a corresponding length of hand-tramming drifts.
2. The ore would be transported underground a greater distance than before. A part of the saving in the length of the hand tram would be lost in the added operation of power haulage.
3. The old shafts must still be maintained for men, timbers, and waste rock, while the main shaft is added to those in use. (The new State law restricting the time of underground labor to 8 hr. from collar to collar, has made it important to have shafts close to the stopes and active workings.)

The advantages were believed to outweigh the disadvantages, and the construction was authorized.

Electric Haulage on Alternate Levels Only

The cost of equipping each level for electric haulage was prohibitive. Instead, each alternate level was chosen, beginning with the 400, making seven in all to the 1,600. By collecting the ore from 15 producing levels to seven haulage levels, the traffic density is increased, although still far below that necessary for really cheap transportation.

By referring again to the haulage map, it will be seen that the 400 level draws ore from the whole of the shallow western end of the mine. It begins in the solid ground under the stopes, passes out to one side of the ore zone and then to the hoisting shaft. It was possible to select old drifts in stable ground for most of the main line. The only new drift of any length was from the Gardner to the Sacramento. Ore is dropped from the 100, 200 and 300 to this level, through transfer chutes, under or near the stopes. In some cases, the haulage cars are collected by hand through short branch drifts to the stopes on the haulage level; in others, when the quantity warrants it, the trolley wire is carried to the stope chutes direct. Of the whole 5,600 ft. of main-line trolley drift on the 400, not more than 1,500 ft. was timbered, and of this only 450 ft. is particularly heavy.

In a similar manner, the 600 level was driven through from the Sacramento to the Gardner, old drifts were used to the Spray shaft, which were extended by new drifts, to reach the ore as it dipped below the 400 level. It underlaps the 400 haulage for some distance. Although this drift passed through the ore zone between the Gardner and Spray, some of the old drifts were in good ground, and the proportion of heavy ground to sound is not much greater than on the 400.

The same plan was carried out for the other levels, each one collecting ore from an area deeper, and farther to the south and east, than that served by the level above. Not very many new drifts were driven particularly for this purpose, and the percentage of main drifts which require repairs is small.

A trolley line on the 200 level to the Uncle Sam shaft was not contemplated in the beginning, but was added afterward. The ore from that shaft and the Southwest country is hauled to a transfer chute near the Holbrook shaft, where it is dropped to the 400. Extending the 400 level to the Uncle Sam to avoid re-handling may be justified at some later date.

To begin with, 7,000 ft. of new drift was driven especially for power haulage, but other drifts have been driven since, and others equipped for the purpose, until at this date, 10.9 miles of track and trolley line are in use or are standing ready. The connecting drifts on the 1,200 and 1,400 levels were warranted as prospects, and many others now used for haulage were driven to find or develop ore. Probably not more than one-quarter of the whole should be charged to the haulage system.

Track is 20-in. Gauge

Twenty-five pound rails on 20-in. gauge are used for the track, with 4 by 6 by 42 in. ties. They have been heavy enough except in a few cases.

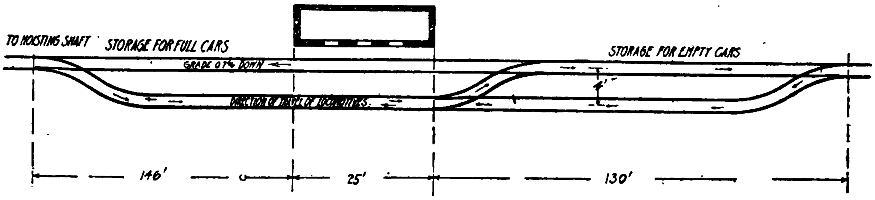

FIG. 2.—TRACK LAYOUT AT LOADING STATION.

The maximum concentrated load is on the 400 level. Locomotives are there used weighing 14,000 lb. The weight of a loaded car rarely exceeds 6,000 lb., although it may be more when loaded with massive sulphide ore. Wet ground on this level caused very heavy track repairs; even when slag was used for ballast, it was churned out from under the ties. To improve this condition, 4,200 ft. of second-hand 40-lb. rails were laid to replace lighter rails. The stiffer rails distributed the load better, and repairs were much reduced.

As a rule, the train crew does not load the ore. The motorman brings empties to the stope or transfer chute, and takes out the full cars standing there, or returns when the others are filled.

In the larger stations, sufficient double track is laid for tail room to hold 20 cars. A standard track layout is shown in Fig. 2. Only one switch, at *A*, is thrown by hand; spring switches direct the locomotives without further attention. At unimportant chutes, less elaborate sidings for car storage are used, or the trains may be made up in the stope drift or on the main line.

At the Sacramento shaft, the tracks are looped to avoid switching. Standard switches on curves of 40 to 80 ft. radius are used. Curves of 25-ft. radius are permitted only in exceptional cases.

Direct Current at 250 Volts Used Underground

Direct current is carried into the mine at 250 volts, through the Sacramento, Gardner, and Holbrook shafts. The feeder cables are cambric insulated, lead covered, and wire armored. The return cables have weather-proof insulation only. The voltage on the west end has been low, for lack of capacity in the power cables. A transformer set has been purchased to be set up at the Holbrook shaft to serve as a booster, but it is not yet in service.

The current underground is carried on 00 B & S American standard grooved trolley wire. The rails are bounded with No. 12064 Type E O. B. rail bonds, length 32 in. center to center, capacity B & S. No. 0, diameter of compression terminals 5/8 in.

Trolley wires are now required by law to be 7 ft. above the track. They are hung from the timbers in timbered drifts, or from 4 by 6 in. cross bars held by vertical bolts wedged into drill holes, when the ground is hard. The old lines were sometimes hung lower, and in such cases were protected by 1 by 6 in. battens placed parallel with the wire, 8 in. apart and projecting 2 in. below the wire. On straight track, the wire is supported every 20 to 25 ft., as very little sag is permissible. Occasionally, the wires must be protected from acid water dripping from the roof. There is also more or less trouble with bonds in wet drifts.

There have been five fatal accidents in which the trolley current has been involved:

No. 1, September, 1908. Pipe-fitter at work on air line above wire. Received shock and fell to track 7 or 8 ft. below. Uncertain whether shock or fall was fatal.

No. 2, December, 1908. Miner repairing drift stood on a truck and fell off. Neck was broken. Thought that shock from trolley wire caused the fall.

No. 3, November, 1909. Loader carried chute bar when switching cars and struck wire beyond protection.

No. 4, May, 1912. Miner climbed over couplings of standing train. Tilted steel on shoulder and touched trolley wire.

No. 5, July, 1914. Miner repairing drift. Ground fell, carrying down trolley wire and pinned him down with wire against his back. May have been killed by the fall. Current was left on trolley wire to light the drift, contrary to rules.

The first two cases were doubtful, as the shocks may not have been sufficiently severe in themselves to cause death. The danger of a current of 250 volts was not realized at once, as it is the lowest in common use, and other mines have not found it so. Other men also in this mine have received shocks without injury. It was supposed to be due to unusual susceptibility on the part of the individuals. It now appears most probable that when shoes are wet, as often happens, with acidulated water, and there is moisture on the hands or body, an unusually good contact may be made, with fatal results.

When this condition was recognized, special precautions were taken to prevent accidental contacts. The wires are high enough to avoid being touched ordinarily. Any reasonable increase in height would not entirely prevent contact when carrying tools, etc., on the shoulder, which is also contrary to rules.

The greatest danger is in repairing drifts or loading ore. No repairs are now permitted in the vicinity of live wires, and the system has been divided to facilitate cutting out the current in sections under repairs, without interfering with ore haulage. At loading stations, the wires were at first protected by boards on each side of the wire. The current is now switched off the wire on each side of the chutes for such a distance that the loader cannot receive a shock while loading, or when switching cars, even in case he should carry a chute bar with him when doing so. When the loading station is on a main line, the current is carried past on an insulated cable, and is only turned on the trolley wire when the train passes and after warning the loader. Current in the wire is indicated by a light.

There are a number of loading stations on each level, and as it takes as long or longer to load, than to make the trip to the shaft, dump, and return, the locomotive usually draws from more than one station. On the 400 level, where the ore is most difficult to handle, and there are many loading stations, they are connected by wire to a signal box containing an annunciator similar to that used in hotels. The loader can thus signal to the motorman when his train is nearly full, and save a trip to a station where the train is not loaded.

Red lights are used in a crude hand-operated block system when there is danger of interference with timber or waste handling, or when there is a second locomotive at work on the same level.

Colored lights are also used to indicate the position in which switches are set, or the location of a place of refuge from passing trains.

Factors Affecting the Choice of Cars

So many of the old drifts were to be used when the system was planned, that it seemed impossible to use large cars.

The first cars bought held 21 cu. ft. They had gable bottoms, with hinged sides, and were equipped with M. C. B. Midget couplers, spring-draft rigging and roller-bearing axles. The latches were to be sprung for dumping at the shaft bins, by a block between the rails.

This type of car was not satisfactory. A large proportion of the ore is too sticky to slide out of the cars, even when the doors are held open. It was customary on the 400 for two men to rock them from side to side on the rails, in order to clear them of ore. The frictional resistance to dumping increases as the size diminishes.

Another disadvantage was their tendency to leak fine ore, particularly if wet.

After using them for some months, a few rocker dump cars were designed and built, of about the same capacity. Spring-draft rigging, automatic couplers and roller bearings were used, as they had been found advantageous. The rollers in the first bearings were solid. For the new cars, an axle was designed to use Hyatt rollers, with special care to make them dirt-proof. They dumped freely, and were otherwise satisfactory.

Since it was necessary to secure a complete new equipment of cars, the question of size was re-opened, and a car of 33 cu. ft. capacity was adopted.

The cost per cubic-foot capacity decreases as cars increase in size. They should cost less for repairs also, in proportion to the ore carried, as the details of latches, etc., are heavier and stronger. Larger net loads can be carried in large cars, as the weight does not increase in proportion to the capacity. The cost of loading and particularly unloading is generally less. For these reasons, the largest car possible should be used.

The size of the cars in this case was limited by the cost of enlarging the drifts, and by the weight which can be conveniently handled by one man.

A large sum of money was spent in widening drifts for the 33-ft. cars. A car enough larger to be of appreciable benefit would have incurred prohibitive expense in widening the 4½ miles of drift then in use, and an added expense for all haulage drifts to be driven in the future.

At important loading stations, the track is laid on a grade to favor the loader in assembling cars. In many cases, the quantity produced will not warrant special sidings, and the cars must be loaded and pushed by hand to the point of assemblage. The 33-cu. ft. car weighs 1,700 lb. Loaded with dry, oxidized ore it weighs about 4,250 lb. When loaded with sulphide, it may weigh as much as 7,500 lb. This car with roller-bearing axles can be readily moved by one man at all transfer chutes, and through branch drifts in most cases, unless the grades are unfavorable. A larger car would quite frequently need two men. The use of cars larger than one man can move would increase the expense of switching by hand, until a car twice as large is used, while that of loading would be practically doubled, since there is only room for one man to work conveniently at a chute.

The rolling-dump type was selected for its ease of dumping sticky ore, and simplicity of construction. Tipples were not seriously considered, as seven would have been required.

The advantages of such refinements as spring-draft rigging and automatic couplers may be open to argument, but a few fingers have probably been saved by the latter.

The roller-bearing axles have been a most unqualified success. They are made in two sizes, and are of as great advantage on small cars as large. The axles on the motor cars are filled with grease once a month; on the hand or mule cars once in two months. It is probable that this time is shorter than necessary, and experiments are being made to determine how long the charge will last. There has been an economy in lubricants accompanied by a very low depreciation charge. A few of the first Hyatt bearing axles, in use since January, 1909, have begun to fail. The cage goes first, which allows the rollers to become displaced and injured. These axles are 1¾ in. in diameter and are used on 22-cu. ft. cars. They are rather light for the service. The larger cars have 2¾-in. axles, which appear to be amply large for the load, and a longer life is expected.

The friction load is very light. Unfortunately, the average mileage per car cannot be known.

TABLE I.—*Wear of Car Wheels, Axles and Roller Bearings*

Size of Car, Cubic Feet	Wheels				Roller Bearings			Axles	
	Date Put in Service	Material	Original Diameter, Inches	Wear in Diameter, Inches	Make	Original Diameter, Inches	Wear in Diameter, Inches	Original Diameter, Inches	Wear in Diameter, Inches
22	Jan., 1909	Manganese steel.	14	0.44	Hyatt	9/16	0.053	1¾	0.013
33	June, 1910	Cast iron	14	0.41	Hyatt	15/16	0.0075	2¾	0.026
33	June, 1910	Cast steel	14	0.375	Hyatt	27/32	0.008	2¾	0.04
33	June, 1910	Cast iron	14	0.34	Solid	27/32	0.023	2¾	0.062

The roller bearings on which the above tests were made were manufactured either five or six years ago. The Hyatt Roller Bearing Co. advises that the rollers are now made of chrome-nickel alloy steel, which has a very much higher elastic limit than the steel used in bearings formerly.

The limits of accuracy in finishing are now approximately 0.002 in. instead of 0.005 in. It is very probable that better results would be obtained with the rollers as they are now made.

Several Types of Cars Used

Several types of cars have been used, and a summary of their characteristics may be of interest.

The original car had a capacity of 12 cu. ft., and was of an ordinary type, dumping forward and also, by swivelling, to the side. It is not worth serious consideration, as it was both light and weak. The older shafts were small, and a car of greater capacity of this construction could not be put on the cages.

Two cars of 16- and 20-ft. capacity each (Fig. 3), dumping only to the side, were recently designed after a car developed by the Old Dominion Copper Mining & Smelting Co. The 16-ft. car will go on the cages in the smaller shafts. The 20-ft. car can only be used in the Gardner, Sacramento, Spray, and the newer prospect shafts. These cars have great capacity for their over-all dimensions. They are of heavy construction. They can be dumped without uncoupling when used in trains. This is an important advantage, as they are frequently hauled in short trains by mules or motor to transfer chutes, or in carrying waste from development work. It is a disadvantage that it can only dump to one side, and care must be taken to see that it comes to the dumping place faced the proper way. It is proposed to replace the 12-ft. cars with these as fast as they wear out. It is unfortunate that two sizes need to be used, but it is believed that the added capacity of the large car makes it worth while.

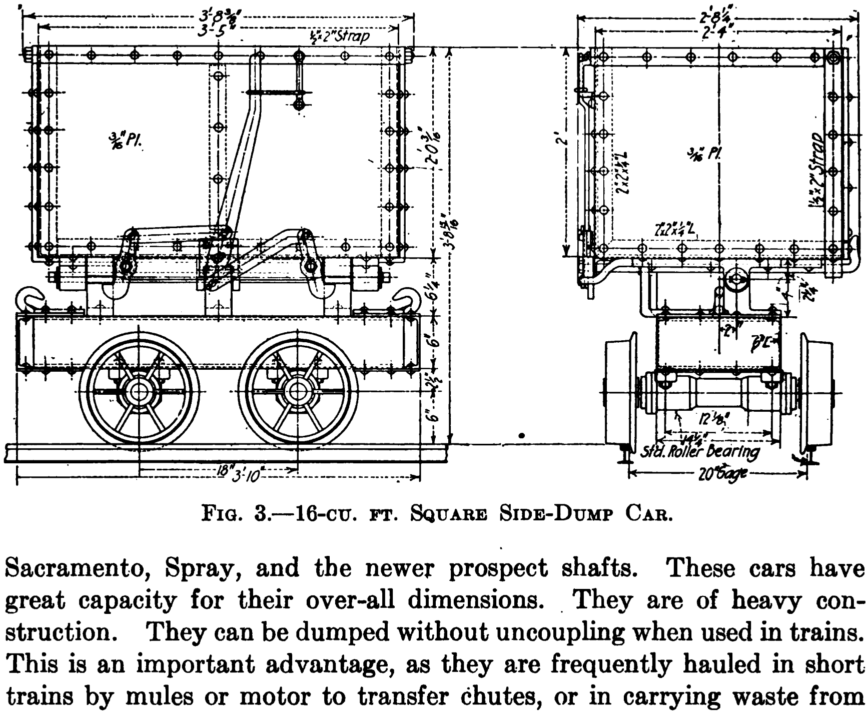

FIG. 3.—16-CU. FT. SQUARE SIDE-DUMP CAR.

Rolling-dump cars of extremely simple construction were designed for use at the North Star Mines, in Grass Valley, Cal., and were introduced at the Copper Queen for surface work. Afterward, they were taken underground, and have proved satisfactory for hand and mule tramming. They are cars reduced to their lowest terms, consisting as they do of a pair of axles with wheels, two beams of "I" or channel section tied together at the ends by two cast-steel supports for the body, which also form bumpers and attachments for connecting in trains. In small sizes, they may be locked by a pin through trunnion and support, if necessary. When properly balanced, they dump with great ease, and do well with sticky ore. As the body forms a tight box, ore is not spilled, even if it is a thin mud. As they dump more easily when well filled, the tendency is to heap them up rather than load them to only a part of their capacity.

They are bulky for their capacity, however, and while this is not important when used exclusively on levels, it is an objection for cage work. They are also high and not so convenient to shovel into. In order to prevent a multiplicity of types, it is probable that the square-bodied side-dump car will eventually supersede these also, as they can only pass through the newest shafts.

The 33-ft. car of this type, which is used for haulage, is too large for general hand or mule tramming, and has most of the advantages and disadvantages of a small car of the same type. It dumps muddy ore more readily than any car which has been tried, but if the ore does not all slide out together, what remains is in the bottom and it is hard to hold in position to dump. The point of support is placed below the center of gravity in order to dump easily, and a lock on the bottom prevents its over-turning in transit. A larger-sized rolling-dump car is not to be recommended. The force necessary to revolve a weight of 2 to 3 tons may be excessive.

In larger sizes also, the objections to the saddle-back car disappear. They dump ore well when they hold 75 to 100 cu. ft. The doors are the only moving parts, the ore is merely permitted to fall out, and, if it does, nothing better can be desired.

Three to 7-Ton Electric Locomotives in Use

There was difficulty at first in securing standard locomotives small enough to go through the drifts; 3-ton Goodman locomotives were selected, their type No. 1600-K. They were small and compact, and were driven by a single motor of 20 to 25 h.p. on one-hour rating. Since, under ordinary conditions, the train crew does no loading, the locomotives were in continuous motion. It gave the motors no time to cool off, armatures burned out, and electric repairs in general were very heavy. About this time, it was decided to use larger cars, the drifts were widened and larger locomotives could be used. Two 6-ton locomotives (6 N-0-2) were then purchased. They were driven by two 8-A Goodman motors of 22 h.p. each on one-hour rating, and have been satisfactory.

The haul on the 400 level is long, with adverse grades for a third of the distance. A still larger locomotive was thought desirable for this level, and two 7-ton machines were designed and built in the mine shop. They are extremely small for their weight and power. The frame is of cast steel. The two Westinghouse No. 79 mining motors, of 30 h.p. each on one-hour rating, were taken second-hand from stock. In the design, everything was sacrificed to compactness and power, and it is not to be recommended for general purposes. The unusually rugged electric equipment has fulfilled every expectation in the elimination of delays and repairs. The tire expense is rather heavy, since the clearance below gears is so small, the tires can only be worn down 1 in. in diameter before replacement.

The small locomotives replaced by the heavy machines have been used to good advantage on light work, either on levels where the ore production is not heavy, or in hauling waste or timber. They were well constructed, serviceable machines when used with the load for which they were designed. Since then, repairs have been general and not confined to any particular feature. Gears and tires make up most of the operating cost. The gears are cut out of special steel blanks on a milling machine at the mine shops. The tires are of forged steel shrunk on a cast-iron center. The treads are re-faced when necessary, and are worn down to a thickness of ¼-in. at times. Manganese-steel tires were tried, but were found to wear very rapidly.

Odometers have only recently been attached, accurate mileage figures therefore cannot be given.

In narrow drifts, it is sometimes impossible to turn the trolley pole, and many were broken in backing up. A trolley pole with a knuckle in the center, which is easier to turn and which can be turned anywhere, has reduced the breakage.

In selecting a locomotive, it should be seen that the motor equipment is large enough for the work to be done; it will save endless trouble, delays and expense.

Power-Rating of Locomotives per Ton on Wheels

Size Locomotive, Tons	One-hour Rating, Horsepower	Horsepower per Ton
3	20	6.6 to 8.3
6	44	7.3
7	60	8.6

Storage-Battery Locomotive Tried

One of the 3-ton locomotives was used for a time in 1910 and 1911 with an Edison storage battery. The results of the running test were presented in a paper by Charles Legrand.[1] The battery was composed of 150 A-6 cells carried in two separate trailing cars.

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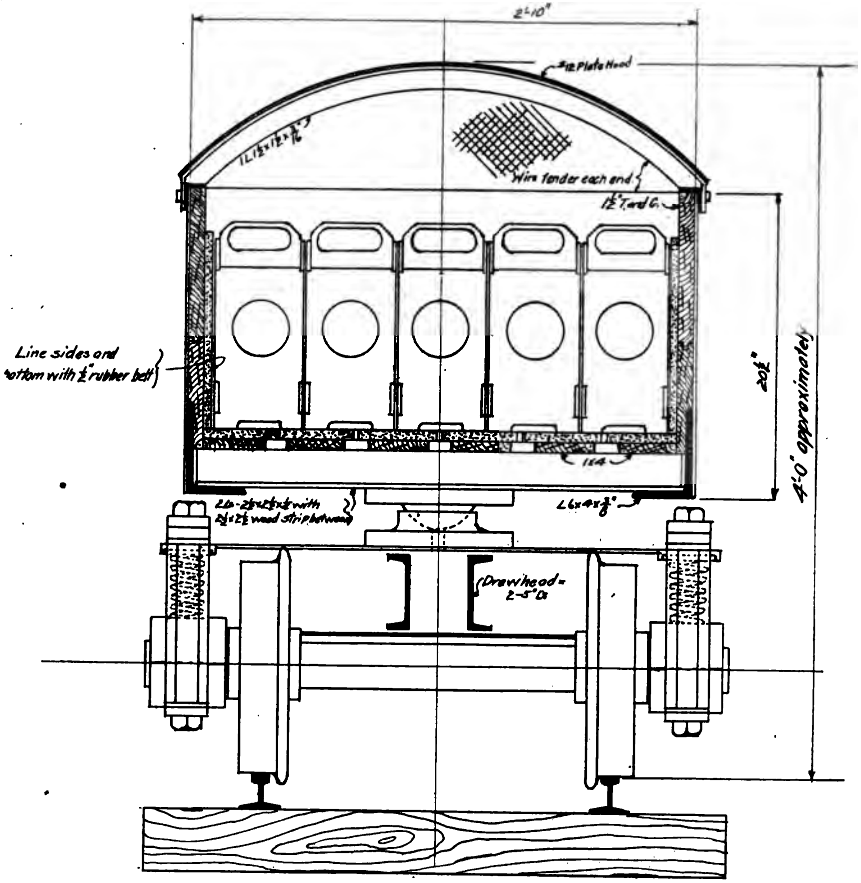
[1] Electric Traction in Mines, *Trans.*, xlviii, 295 (1914).

Moisture from the air accumulated on the cells, to which was added a small proportion of electrolyte as a result of spillage or gassing while being charged, and a number of cells were lost by short circuiting.

In 14 months, 21 cells were burned out. The whole battery was then returned to the factory, and replaced by another with different crating to provide better insulation. A new battery car provided with springs (Fig. 4) was built large enough to contain them all. It weighed 2,400 lb., and the battery 3,000 lb. The cells were occasionaly wiped dry, painted and coated with vaseline. The nickel cell walls of only two cells were burned through in the 7½ months of its service. The cells at the ends of the cars were those usually lost.



Cross-Section of second Car

FIG. 4.—UNDERGROUND MOTOR STORAGE-BATTERY CAR.

If the batteries could have been kept perfectly dry, or if the voltage had been lower, there should have been little or no difficulty from short circuiting.

When idle and fully charged, the voltage was 213, after travelling about 8 miles, one-half under load, the voltage dropped to about 200. When the voltage dropped still lower, the speed of the locomotive decreased and the work went slowly.

Ore was drawn during the day and the battery was charged at night, by current from a 20-h.p. motor-generator set taking power from the alternating-current circuit. It was automatically cut out when charging was complete.

The production from the area served by this locomotive gradually increased until its capacity was exceeded. It was then thought better to lay a wire than to get another battery, and the use of the storage battery was discontinued. The cost of the locomotive was approximately $1,300, that of the battery about $3,000.

The operating costs were good in spite of light rails and sharp curves. No information on normal depreciation can be given.

Favorable conditions for motor and battery are opposed. In order to protect the battery, the motor must not be too large. And yet it has been found that motor repairs are reduced by making them powerful enough to slip the wheels of the locomotive readily. The added weight of the battery gives them good traction, and there is always a temptation to pile on the load. By incorporating the battery with the locomotive, they may be better suited to each other and the work to be done.

The capacity is not very great. The cost of battery, charging station, etc. is to be balanced against that of trolley wire and rail bonding. The charge for power is so small that the greater consumption of power-house current is of no particular consequence.

The battery is now set up in the power house to excite the fields of the alternating-current generators when starting. They replaced a small steam-generator set, and have been very useful.

Several Designs for Stope and Transfer Chutes

When the stopes are on a haulage level, only the regular stope chutes are used, and the cars are brought to them by the locomotives, if the tonnage is of sufficient importance, or by hand from some nearby point where they may be re-assembled into a train to be hauled to the shaft. If the ore comes from the level above, it is dropped through a transfer chute to the haulage level. This may be an old stope chute or it may be a special raise into which the ore from a considerable area is collected.

For the larger bodies, a chute is usually provided close to the stopes, even if a special drift must be brought in from the main line. In a top slice, the main chute is brought up to the top floor and the ore is trammed from the working faces to the haulage chute direct. In the Dividend slice, the ore being too soft for raises to be maintained through it, they are placed in the country rock outside, and the ore will be trammed a longer distance on the working floor of the slice, instead of to a central raise within the ore.

In the Howell slice, the ore mined above the 600 is dropped to the 1,000, as it is economically impossible to maintain a trolley drift through the ore zone on the 800. In this case, the ore is trammed again on the 900, as there is not sufficient to justify a special raise clear through.

Table II.—Loading Cost for Levels in 1914

Level	Cents per Ton
400	6.4
600	2.4
800	3.4
1,000	3.5
1,200	1.8
1,400	2.8
1,600	3.2

The 400-level costs for loading are highest, since the greatest proportion of clay ore originates in the area tributary to that level. The ore in the Dividend stopes is a very tenacious clay, and a cost of 10c. per ton for loading has not been uncommon nor excessive. The difficulties of loading have directed particular attention to the development of a better chute than that formerly employed.

A rectangular vertical chute of small section, with a bottom sloping down to a door in the front, is the worst possible for sticky ores. The small section increases side friction and favors arching. The weight of the column of ore packs it tightly into the wedge-shaped section at the bottom, particularly when it falls as a mass after hanging up. Only a part of its weight acts as a force to push it horizontally through the door. By shortening and widening the column of ore, the weight per square foot on the bottom layer is lessened, and there is less possibility of arching. By putting the door underneath instead of at the side, the full weight of the ore acts to force it through. These principles were used in developing a chute which has been found to be of great advantage. The doors have been put underneath the bin in only one case, as it is seldom necessary.

From the sketch of a standard transfer chute (Fig. 5), it will be seen that the storage is all at the bottom. The ore falls through the chimney and strikes a baffle, its downward velocity is checked and it falls lightly on the heap below. The pile of ore is not allowed to extend up into the raise, which serves only as a passage way. The storage space is long enough to let it spread into a loose pile, there is little tendency to pack, and it cannot readily arch. Below, there are several doors. If, when loading, the ore hangs up over one door, the car is moved to the next, which will break the arch, or if that door does not, to the third. The bin itself is usually placed in solid ground and requires no timber except in the bottom. The narrow chute above is only lined in heavy ground, and then usually with concrete. Its small diameter, from 3 to 3½ ft., brings this expense within reason.

Seven chutes have been lined with concrete. One which has been in use for 20 months, needs re-lining, due to wear by sharp ore. This chute would have required heavy repairs if timbered, both to provide against outside pressure and to replace the lining. In re-lining, a thicker wall of concrete will be used. Other chutes are only slightly worn.

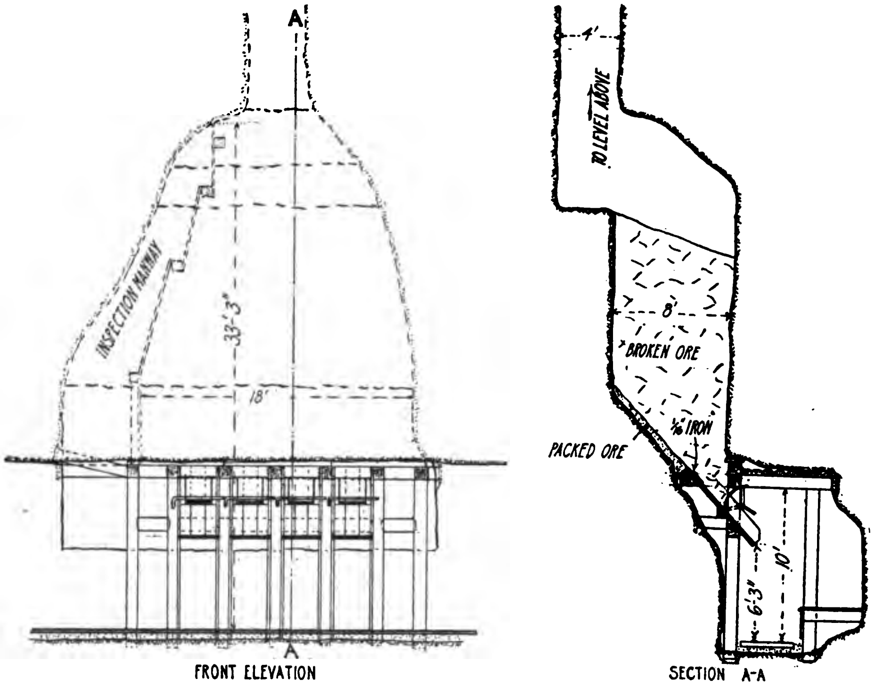


Fig. 5.—Standard Haulage Chute. 90 Tons Capacity.

An elaborate storage bin, shown in Fig. 6, was built on the 400 level to serve a large aluminous orebody which has been developed, but not yet mined. It was described by Joseph P. Hodgson, Mine Superintendent of the Copper Queen mines, in a paper read at the Salt Lake meeting of the Institute.[2] A large tonnage of ore is tributary to it, which is so sticky that extreme measures were considered necessary.

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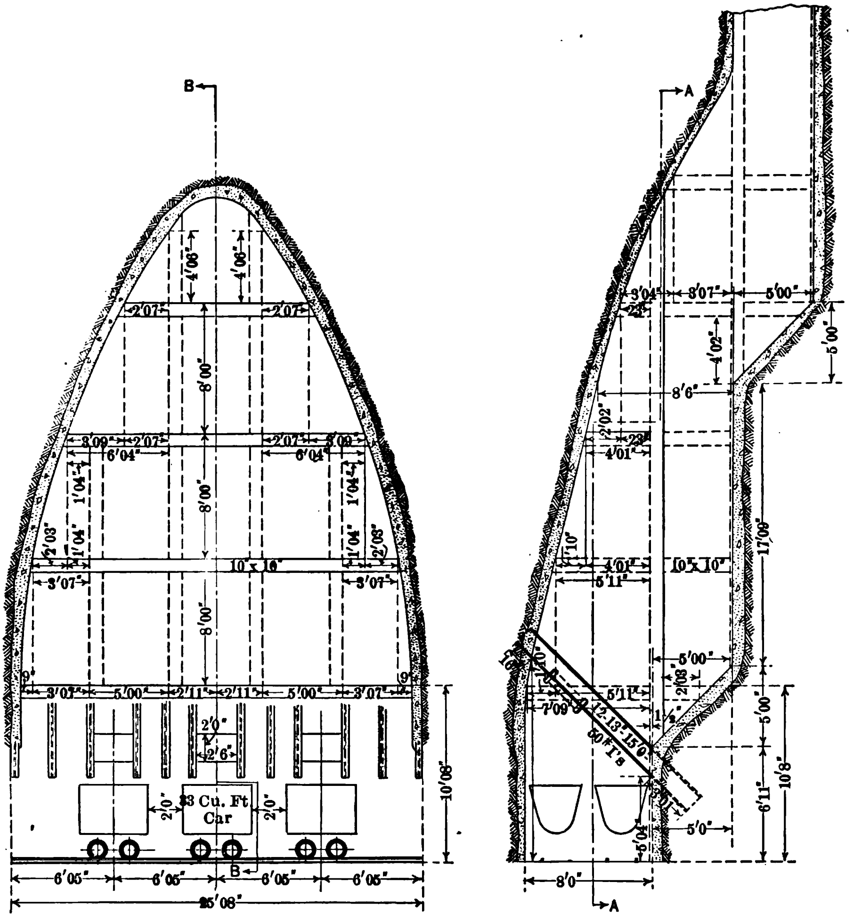
[2] Mining Methods at the Copper Queen Mines, *Trans.*, xlix, 316 (1914).

This is the case in which the doors are put underneath the center of the pile of ore. There is an opening from one end to allow the loader to climb up on the pile, if necessary, for inspection, or to work the ore down. A ¾-in. pipe is laid in the concrete lining, to serve as a speaking tube between the loader and the trammer above.

The ore from this stope had formerly been dropped from the 200 to the 400 through a standard, timbered chute. No attempt was made to use the chute for storage, and the door at the 400 was kept open by the loader, but other men were required above to keep the chute clear and the ore moving.



SECTION A-A

SECTION B-B

FIG. 6.—CZAR ORE POCKET.

It was found that the whole capacity of the storage bin could not be used, the weight of the ore packing it too hard; but if the quantity does not exceed 50 or 60 tons, the loading is rapid, and there has been no accumulation of ore in the concrete chimney at any time. In rebuilding it, the storage chamber would not be made so high.

The timbered chute shown in Fig. 7 is sometimes used successfully for stopes. The half bulkheads partly support the ore, and it does not pack so badly in the bottom. It was used by the Cananea Consolidated Copper Co.

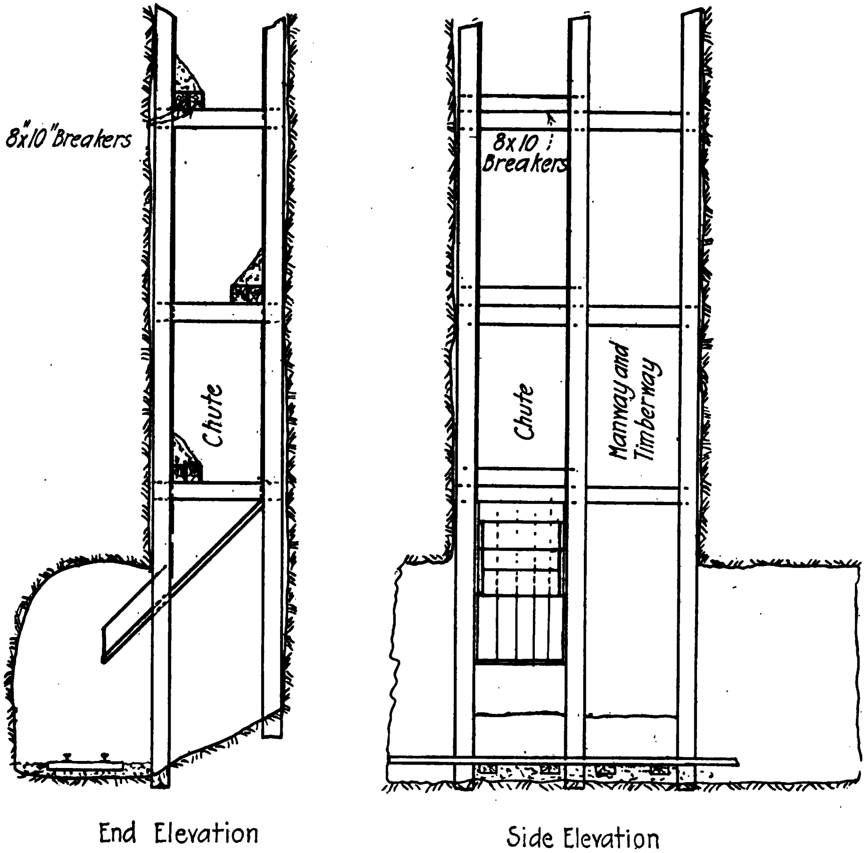


Fig. 7.—Timbered Chute.

Sacramento Selected for Hoisting Shaft

The greatest month's production prior to July, 1907, had been, 55,347 dry tons, in January, 1906, or 1,910 tons per working day. It was not exceeded until March, 1912. The desired capacity of the plant was assumed to be 60,000 tons per month, or about 2,150 tons per working day, with a reasonable margin for emergencies. At that time, the mine worked alternate Sundays, and three shifts were run in tramming and hoisting.

The Sacramento shaft had been sunk for general development purposes, and had been located within the porphyry intrusion, to avoid the ore zone and its possible movement. It was selected for the hoisting shaft since it was in a central position and convenient to the main line of the railroad.

It was 928 ft. deep to the 1,000 level, and had three compartments 4 ft. 6 in. by 5 ft., and a half compartment for air pipes, etc. Since then, it has been sunk to the 1,700 level and widened to five compartments, making two for skips, two for cages and a large one 5 ft. by 5 ft. for pipes and cables.

It was at once decided to use skips.

The size of the compartments restricted the cross-section of the skip to 4 ft. 2 in. by 3 ft. 3 in. inside. The effective capacity fixed upon was 60 cu. ft. It was thought that one of large capacity would be too long, and that the bottom layer of ore might be packed so hard by the fall into the skip and the weight of a high column of ore upon it, that it might stick. This has sometimes been the case with the size adopted. The average skip load in April, 1915, was 8,117 lb. of wet, or 7,459 lb. of dry ore. The moisture, 8.11 per cent., is the lowest in years, the average would be nearer 12 per cent.

There would be no difficulty in raising much more than the quantity of ore assumed, when it runs freely, even with small skips. When it does not, large skips are of no particular advantage.

The uncertain factor was the rate at which ore could be drawn from the storage bins, which are necessary when skips are used. The experience with bins had not been particularly successful. Attention was therefore concentrated upon providing facilities for rapid loading, and to give loaders the maximum time for work at the loading door.

The size of the skips, power and efficiency of the hoist and economy of rope wear are interdependent. Conditions favorable to one are usually opposed to efficiency in the other.

A larger skip requires a more powerful and expensive engine, the efficiency of which is less, due to less continuous hoisting for the assumed tonnage, and greater condensation. There might be a small saving in rope expense, as the skips would make fewer trips.

Maximum efficiency in ropes would be obtained by larger drums, or drums long enough to avoid two layers. Larger drums require larger cylinders and again an engine more costly and less efficient. There is no great advantage in the use of larger drums unless they are large enough to hold the rope in one layer.

The capacity of the skips, the proportions of the engine, and diameter of the ropes were arrived at by one of those compromises, which must be so frequently made in balancing an advantage against its complementary disadvantage.

Reports of the steam economy of compound engines operating in South Africa had been received, and after thorough investigation by John Langton, Consulting Mechanical Engineer for the company, a duplex, tandem-compound, condensing, first-motion engine was purchased from the Nordberg Manufacturing Co. The conditions were not the most favorable for a compound condensing engine, as the depth of the center of gravity of production in 1906 was only 570 ft., but it was expected that the depth of hoisting would increase. The center of gravity of production is now 970 ft. below the collar.

The plant is driven by steam at 150 lb. power-house pressure, transmitted 619 ft. through a 5-in. main. The steam leaves the power house with 75° F. of superheat. The engine can lift an unbalanced load of 17,000 lb. Its speed is rated at 2,000 ft. per minute. The reels hold 2,100 ft. of $1\frac{1}{4}$-in. rope in two layers. The cylinders are 18 in. and 28 in. in diameter by 48-in. stroke.

Most of the details are of standard Nordberg design, but there are three unusual features:

The operation of the auxiliaries.

The regulation of pressure on the low-pressure cylinders.

The condensing system.

Auxiliaries Actuated by Oil under 135 *lb. Pressure.*

When steam is used by the auxiliaries, which operate brakes, clutches, etc., it constitutes an important percentage of the total power used by the engine. The condensation is great and continuous, and since the controlling valves can have no lap, but must be set "line and line," it is impossible to avoid leakage as well. The steam consumption by auxiliaries in a simple engine running at capacity is rarely less than 14 per cent. of the total. If the engine runs intermittently, the proportion used by the auxiliaries is correspondingly greater. In a more efficient engine, the steam for auxiliaries remains the same and is also a greater proportion of the whole.

In this engine, throttle, clutches, brakes and reverse are actuated by oil under a pressure of 135 lb. and controlled by poppet valves. The oil discharged from the auxiliaries is pumped back into an accumulator by a triplex pump, with plungers $2\frac{7}{16}$ in. in diameter by 4-in. stroke, which are driven by eccentrics on the valve-gear shaft. When the accumulator plunger is raised to its highest position, the oil is bypassed to a reservoir. When hoisting from the upper levels, the pump is too small and a relay steam pump which cuts in when the accumulator plunger is below a certain point runs most of the time. This method is entirely satisfactory, and the engine control is sensitive, positive, and efficient.

The operating levers are arranged in a standard manner (Fig. 8) first used on the Frazer and Chalmers 20 by 48 in. engine at the Spray shaft. All the large hoists are controlled by levers on this plan. There is a great advantage in having all engine levers in relatively the same position, acting by the same motion. Engineers can be transferred from one hoist to another without danger of confusion.

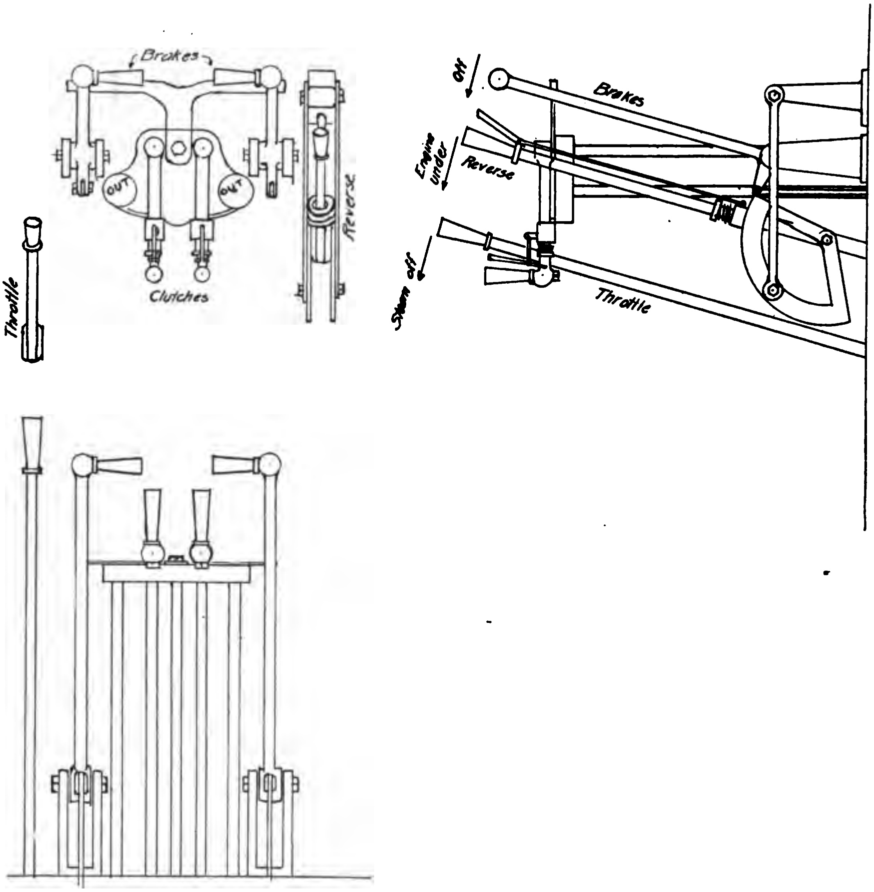


FIG. 8.—TYPICAL HOIST-LEVER OPERATING SYSTEM.

Pressure on Low-pressure Cylinder Automatically Regulated

A large steam-jacketed receiver is placed between the high- and low-pressure cylinders, in which pressure is maintained at 20 lb. by an automatic valve, through which steam is bypassed from the high-pressure main to the receiver, when the throttle is closed. Thus, the engine has pressure on all cylinders when starting, and picks up smoothly and quickly instead of needing a stroke or two to reach a working pressure in the low-pressure cylinders. When the throttle is opened, the bypass is shut off and the pressure in the receiver drops to that dependent on the load, speed, etc. When it is desired to hoist with great speed, quicker acceleration may be obtained by adjusting the reducing valve to raise the receiver pressure.

Condenser Operates under Severe Conditions

The engine has Corliss valves, with automatic cutoff regulated by a governor. Severe conditions are imposed on the condenser by the irregular use of steam. During the first two or three strokes, while the engine is starting the load, steam is taken nearly full stroke, while at the end of the trip the cutoff may be at 1/10 stroke. In order to condense this sudden rush of steam, a large reservoir of cold water is provided in the "Rain Type" Nordberg condenser cylinder, into which the steam is exhausted, by a series of shallow flat trays filled with water. Constant streams of water are poured into them which over-flow, rain down to the bottom of the cylinder and finally run off by gravity against the vacuum, to the cooling tower, from which it is re-pumped. A 900-gal. rotary pump is used for circulating the water and a 19 by 18 in. vacuum pump, both belted to 20-h.p. 220-volt alternating-current motors which run at constant speed. The temperature of the water at present (June, 1915), is 84° F. when entering and 96° F. when leaving the condenser. The barometer stands at approximately 25 in. The vacuum drops to 16½ or 17 in. at the beginning of a trip, but rises to 20 or 20½ in. The man engine also runs condensing and exhausts into the same system, which was increased in capacity by large pumps and a second condenser when the man engine was added.

Hoisting Speed 1,300 *to* 1,600 *ft. per Minute*

In order to insure continuity of operation under all conditions, the low-pressure cylinders are built to stand high-pressure steam. The receiver pressure can be built up to 150 lb. and the engine operated as a simple engine with cylinders 28 by 48 in. stroke. This makes it an engine of great power for emergency use, as for instance, hoisting unbalanced, but at a sacrifice of efficiency.

The engine governor is set for a rope speed of 1,300 ft. per minute when hoisting from the 1000 level or a level above, but by shifting the governor belt to another pulley, a speed of 1,600 ft. is used for greater depths. When production is increased, both speeds are raised. It is preferred to run the hoist at as regular and slow a rate as will conveniently hoist the ore.

Round Rope on Grooved Drums Now Used

Round ropes, 1¼ in. in diameter, in two laps are used on reels, which are 7 ft. in diameter by 5 ft. face. The drums were made without grooving at first, as it was uncertain what ropes were best, a high-grade rope 1⅛ in. in diameter or 1¼ in. rope of less tensile strength. The ropes began to cut into the drum in an irregular curve. They were grooved in April, 1914, to a depth of 5⁄16 in., leaving the diameter of the drums at the bottom of the grooves 83⅜ in.

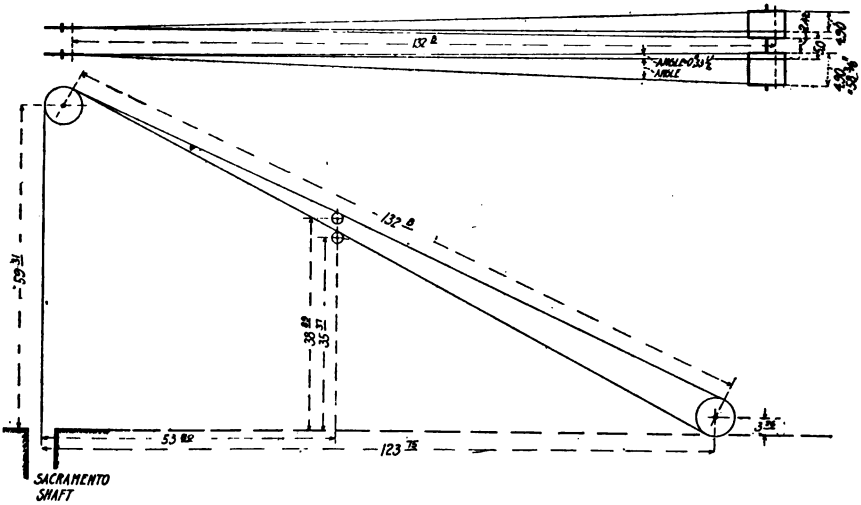


FIG. 9.—LEAD AND FLEET ANGLES, SACRAMENTO ROPES.

Fig. 9 shows the relative positions of drums and sheaves to shaft. The ground back of the shaft was cut out as far as practicable to get a reasonably long lead from the sheaves.

The fleet angles are unequal, and from the sheave line to the outside of the drum is 1° 33′. This would be considered excessive in certain districts, the copper country of Michigan for instance, but accepted practice in other places permits an angle of 1° 55′. The original attachments of the ropes were at the outside, or clutch side, of the drums. There was sufficient angularity in the pull from the sheave to draw the laps apart and leave openings into which the second layer of rope dropped. The winding was irregular and rough, and the wear excessive. Other sockets for rope attachment were cut on the inside of the drums, and the trouble ceased. The angular pull tends to hold the laps together instead of separating them. When the drums were grooved, the ropes were again attached at the outside, since the active part of the rope would be nearest the sheave line, and the fleet angle smaller. There is heavy side friction on the rope when the last laps of the second layer are wound on. If attached at the inside of the drum, the grooving would carry it to the outside without so much friction as if winding against a rope. On the second layer, the pull of the rope would again tend to separate the laps and reduce the side friction of rope against rope. Soft-steel wedge-shaped fillers lift the rope out of the inside groove, high enough to wind back on the first layer.

No difference has been noticed in the wear of the under or over wind. If there is a reasonable distance between sheave and drum, there should be none.

The sheaves are 7 ft. in diameter, and are lined with leather. Wood lining broke down rapidly. Cast-steel idlers support the ropes between sheaves and drums to reduce slapping. They are cast thin and the flanges machined to make them still thinner and lighter.

The ropes are attached to the skips by thimbles and six Crosby rope clamps. Once every four weeks, 6 or 8 ft. is cut off the end of each rope to change the points of maximum wear.

The shaft is usually dry. In the wet season, a little acid water seeps in, but no corrosion has been discovered.

A few years ago, Mr. Duncan, Agent of the Cleveland-Cliffs Iron Co., kindly furnished rope records from the Lake shaft hoist, which has 7-ft. drums, 9-ft. sheaves and uses 3½-ton skips, but with only one layer of rope on the drum.

TABLE III.—*Data on Hoisting Ropes*

	Cleveland-Cliffs	Copper Queen
Diameter of drum, feet	7	7
Diameter of sheave, feet	9	7
Length of rope, feet	1,800	4,000 (2 ropes)
Diameter of rope, inches	1¼	1¼
Weight of skip, pounds	4,000	6,350
Weight of ore, pounds	7,000	7,400 (dry)
Weight of rope, maximum, pounds	1,347	3,880
Ore and rock hoisted, tons	700,000	395,569 (dry)
Depth of average hoist, feet	550	970
Depth of maximum hoist, feet	550	1,550

The Copper Queen rope averages do not include those ropes which were obviously unsuited to the conditions.

The ratio of rope to drum diameter is 1:67.2. The experience of the Cleveland-Cliffs Iron Co. indicates that good service may be obtained with that ratio.

The rope cost on the Copper Queen ropes in the above comparison was 0.3 c. per dry ton. The ore contained about 11 per cent. moisture during the time covered by the records.

From the above comparison, the life of a rope is increased 75 per cent. by the use of one lap instead of two, and the cost for ropes at the Sacramento shaft would have been in consequence, 0.17 c. instead of 0.3 c. This would represent a saving in rope in an average year of about $1,000. In order to obtain this saving, the cost of the engine would have been approximately $15,000 greater, and it would be less efficient. A geared hoist would avoid some of these objections, but would introduce others.

The greatest tonnage, 549,324 dry tons, was hoisted by a pair of flattened strand ropes with Lang lay. The cost was 0.277 c. per dry ton. The increase in wearing surface provided by flattened strands is very noticeable. In winding on a smooth drum, the contact between two consecutive laps is made by a rubbing motion. In the ordinary lay, the outside wires which touch are parallel with the length of the rope at the point of contact, and the rough surfaces of the projecting strands bear against each other when the rope comes down against the last wrap. With the Lang lay, they are inclined to the line of the rope and each other, and the friction is reduced.

These conditions are of paramount importance when two layers are wound on a drum, since the bearing of the second lap is entirely rope against rope, and there is at the best a good deal of sliding of one rope on another. The external wear only need be considered, internal wear is of no consequence in comparison. Efficient lubrication on the outside of the rope is of great benefit.

This question is treated in some detail, because it was found very difficult to obtain information on the effect of small drums and two layers of rope on the cost of hoisting. It was of course recognized that the wear would be increased, but to what degree it would increase the cost could not be ascertained.

Underground Storage Bins Have Loading Hoppers

The storage bins at the haulage levels were at first made of standard design. The only attempt to avoid the difficulties formerly experienced with bins was to make them comparatively small to avoid pressure.

Attention was particularly directed toward giving the loader as much time as possible to poke the chute. This was done by putting measuring hoppers holding one skip load each below the bins, which could be loaded at any time while the skip was in motion and tripped into the skip when it reached the loading position. When hoisting in balance at the maximum rate expected, a skip per minute, and allowing 15 sec. for tripping the hopper and ringing off the skip, each loader has 1¾ min. to load the hopper.

When starting the plant, it was found at once that the customary bin was not satisfactory. Not more than half the requisite speed could be attained. The 800-level bin was altered to one of the present type, but of small size, 40 tons capacity. As soon as it was found successful, the 400 and 600 bins were rebuilt but on a larger scale. Their capacity is 90 and 100 tons. The three lower bins are about the same size.

The bin as reconstructed consists of two parts, an upper chamber for storage and a lower part open in front which acts as a slide to the gates. The horizontal opening at the bottom of the storage chamber is 4 by 8 ft., large enough to prevent arching, but small enough to carry the weight of the ore in storage without permitting it to exert much pressure on the ore resting against the gate. The front, immediately below the contraction, is open for an added relief against pressure as the ore is free to take its natural slope.

This principle has since been used in a 260-ton bin for the 400 level, which is shown in Figs. 10 and 11, together with a standard wooden bin. In this case, the excavation was made circular and lined with concrete lightly reinforced. The conical bottom is armored with rails set in the concrete. It has been found necessary to protect the concrete against erosion when hard ore is stored, but only where the ore strikes on rebounding from the inclined plane at the bottom of the chimney.

It was not known at first how large the hole in the bottom of the cone should be. It was accordingly made 6 ft. in diameter, with the expectation of enlarging it if necessary. It has since been cut out to 7 ft. 4 in., which appears to be the correct proportion to protect the ore below from excessive weight and still prevent arching. This storage bin, although much larger than the others, handles muddy ore equally well.

As finally constructed, the large bin was built on the opposite side of the shaft from that shown in the drawing.

Arc doors are generally used, 30 by 16 in. in size, with good results. The slight extra labor of operation is balanced against the higher cost for installation and maintenance of power doors, which are only used at the large concrete bin at the 400 level.

Below the doors are the loading hoppers (Fig. 12) which hold 58 cu. ft. each, or one skip load. The flat counter-weighted bottom is hinged at the rear and held shut by tension bars on each side extending up to a shaft which is turned by hand levers. It dumps quickly and locks positively, as the tension bars are turned past the center of the shaft. The chute beneath, which carries the ore to the skip, is very steep, to prevent sticking.

The operation of loading is very rapid under ordinary conditions.

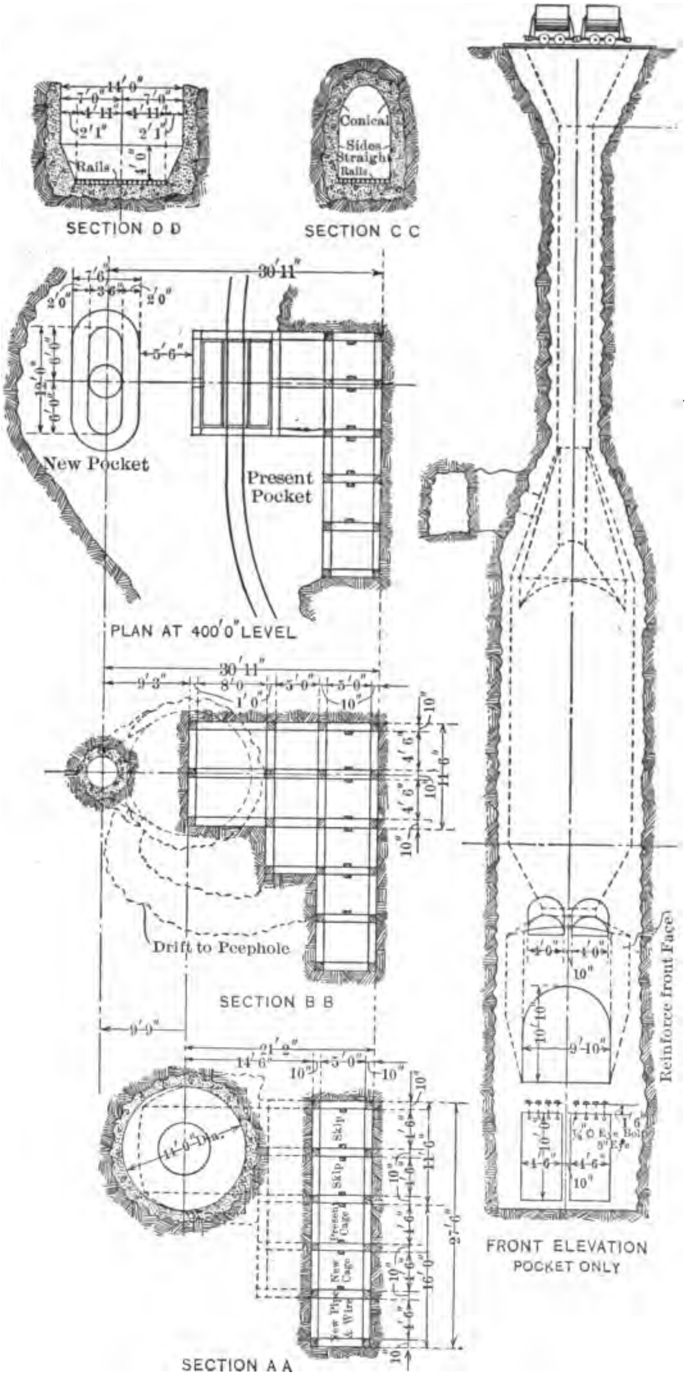


Fig. 10.—260-ton Ore Pocket, 400 Level.

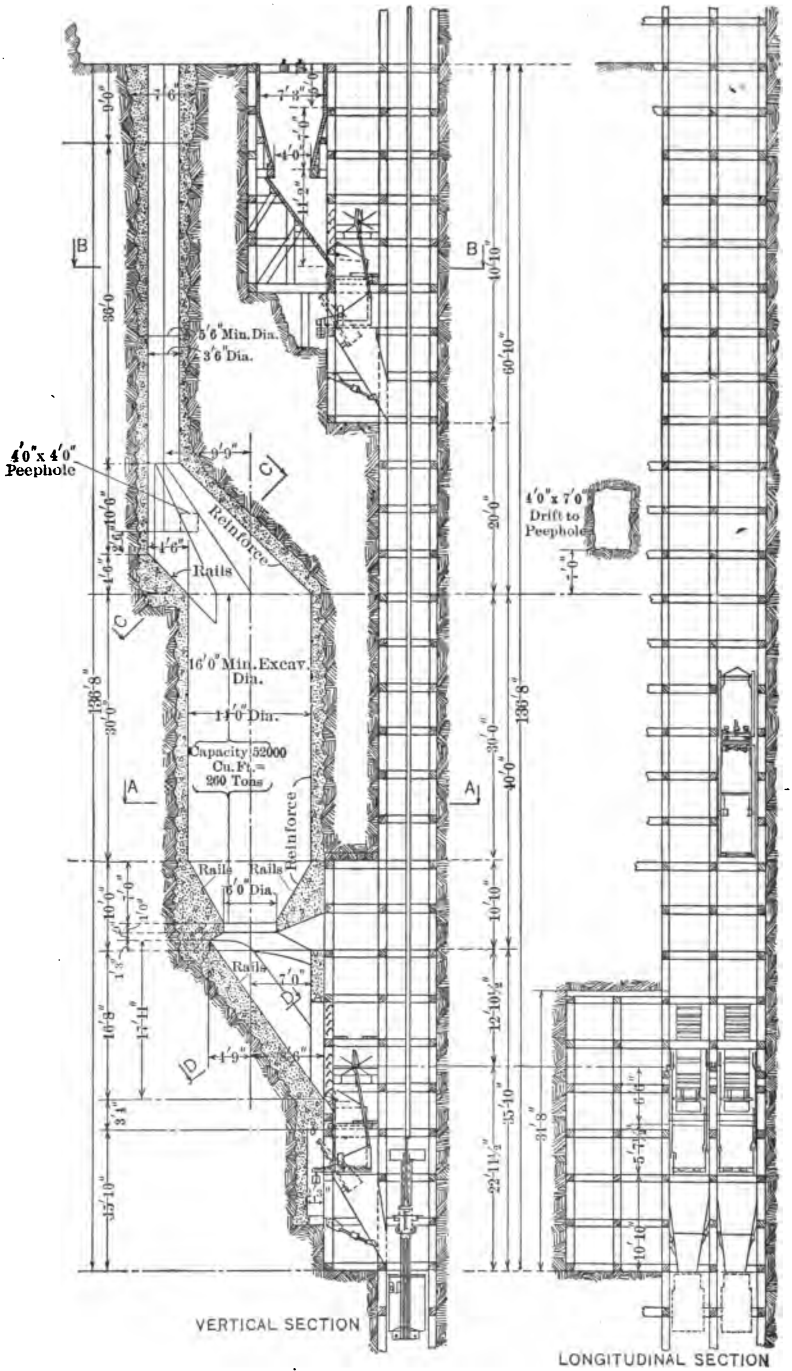


Fig. 11.—260-ton Ore Pocket, 400 Level.

The time required to dump the ore into the skip and ring it off rarely exceeds 12 sec., and frequently not more than 10 sec.

The best day's record is 3,899 wet tons in three shifts, or 3,467 dry tons. An average has been maintained for several shifts at the rate of 433 skips per shift. When the 30 min. for lunch and the time estimated for changing levels are deducted, it makes a hoisting rate of 1.1 skip per minute. Two loaders are required. When there is much muddy ore, or when hoisting very rapidly, a third man is sometimes used. There are still occasions when, for short periods, not more than 20 or 30 skips per hour can be hoisted.

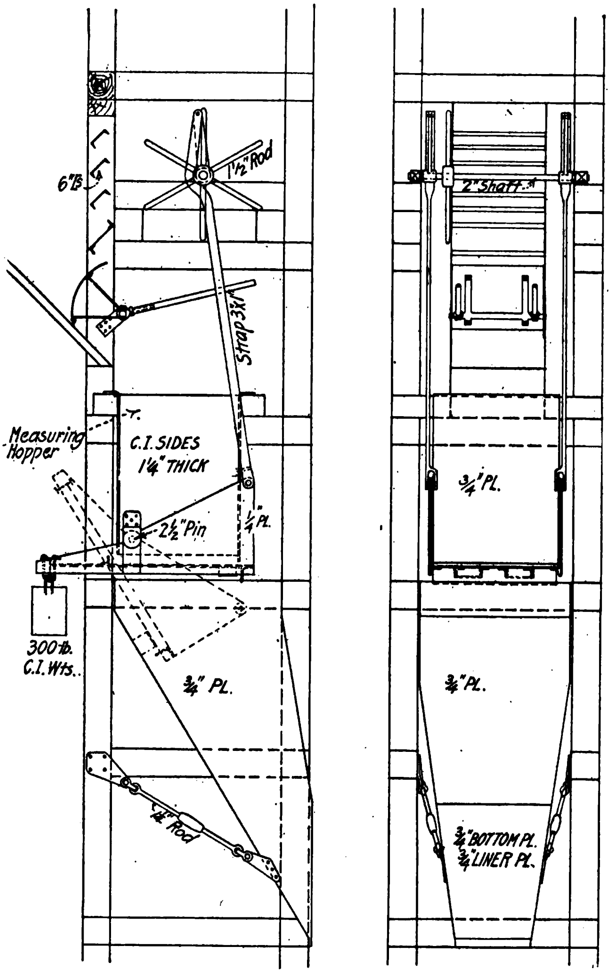


Fig. 12.—Sacramento Measuring Hopper.

The construction of skips and loading mechanism is heavy and strong. There have been few delays and light repairs.

There has been trouble with the spillage of ore while loading. The chute through which the ore reaches the skip is so steep that the opening is 40 in. high. Even if the skip is held only 3 in. below the bottom of the chute when spotted for loading, there is an opportunity for the ore to spread in falling, and for some of it to fall on each side of the skip rather than in it.

A screen was attached to the side bars above the skip, composed of light plates, inclosing three sides but open on the loading and dumping side. It reached down to the top of the skip, but was spaced far enough outside, so that the skip could turn through it to dump. Since there was a certain amount of danger in adding this complication to the skip, and the spillage was not greatly reduced, it was removed.

The spilled ore falls into a pocket at the bottom of the shaft, from which it is loaded into cars and hoisted on the cage to the 16th level bin. It amounts to about 17 lb. per skip, or 0.21 per cent. of the ore hoisted. It results partly from ore falling outside the skip when loading, partly from the ore which hangs up in the slide below the loading hopper and which falls after the skip is hoisted and partly from overloading. If the skip does not dump clean, and it is not noticed, there may be enough remaining to make an overload when the next charge is tripped into it. As a result of the spill, there has been heavy wear on the timber in the hoisting compartments.

The skips had originally only 70 cu. ft. capacity, and not sufficient margin to prevent an occasional overload. They were rebuilt and made to hold 90 cu. ft.

The skip is of standard Kimberly type, but built heavily. The dumping guides on the surface are formed of angles and heavy steel castings. As the castings wear, thin lining plates are riveted on.

Ore Loaded into Railroad Cars by Belt Conveyors and Trippers

The ore is dumped into a small bin holding about 12 tons, from which it is carried to the railroad track below, and loaded into cars for shipment to the reduction works. The arrangement of the conveyor plant is shown in Figs. 13 and 14.

A 36-in. steel pan conveyor forms the bottom of the bin, and in transporting the ore to an inclined belt equalizes the load. The receiving bin was made small to avoid a great accumulation of ore and consequent packing. It is now apparent that a larger bin would have been permissible and desirable.

The ore falls from the pan conveyor into a belt loader which breaks its fall and feeds it to the belt in the direction of belt travel (Figs. 15 and 16).

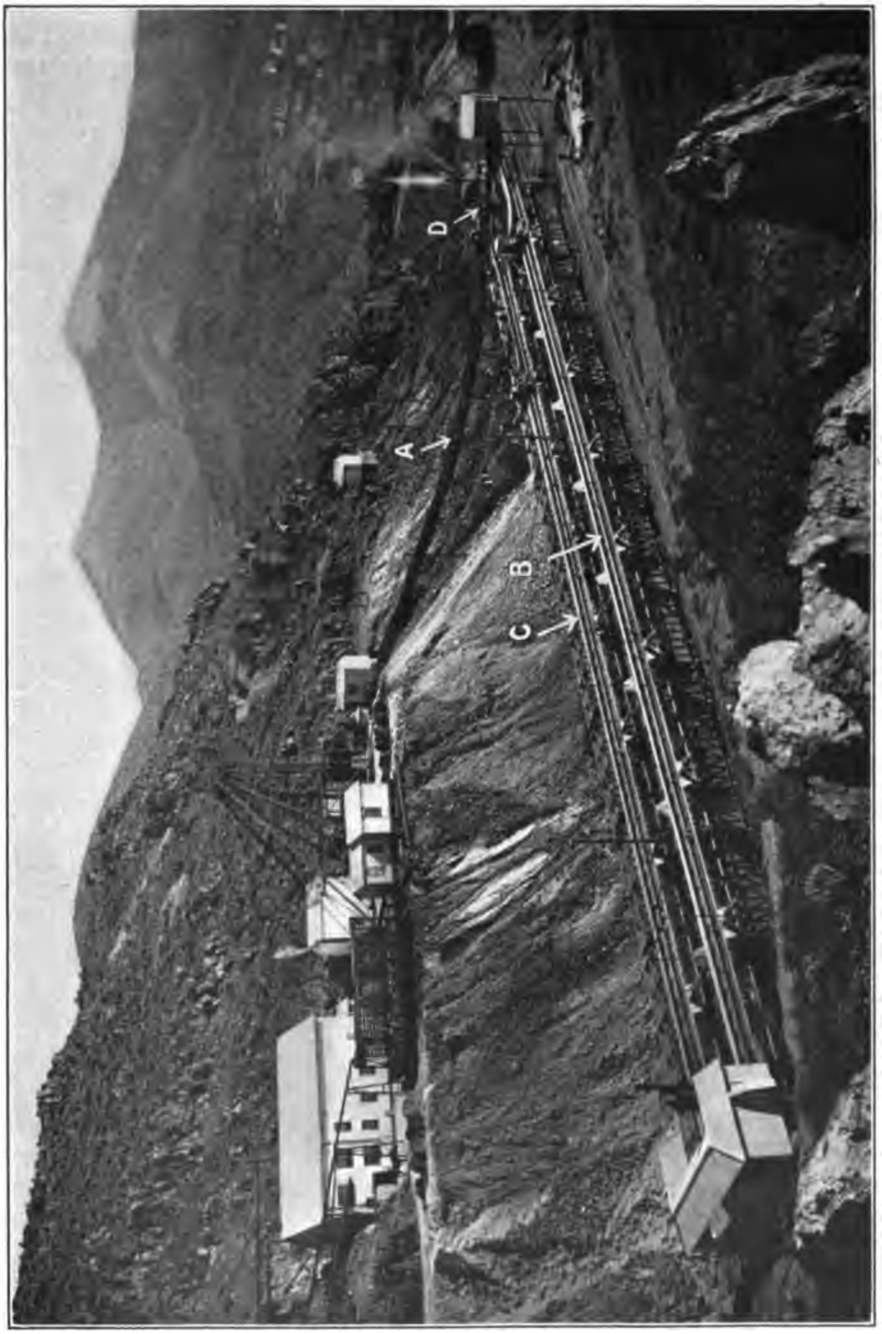


FIG. 13.—SACRAMENTO SHAFT LOADING PLANT IN 1909.

The V-shaped opening at the bottom of the pan allows the fine material to reach the belt first and form a bed for the coarser. The sides of the "V" are movable, and may be spread apart to widen the opening when sticky ore is hoisted.

Great difficulty was encountered in obtaining a feeder which would not clog with clayey ores. Even as it is, a solid mass of clay has accumulated above the plunger, the full width of the feeder, and several feet high, with the plunger still running ineffectively behind.

When the conveyor plant was first built, the ore fell from the pan conveyor upon the unprotected belt, which travelled on a slope of 15°. Boulders of ore sometimes bounded down the incline before they came to rest on a bed of finer material. When the feeder was put in, the upper 18 ft. of the belt was depressed to a horizontal position, and the bed of ore is well settled before it reaches the incline.

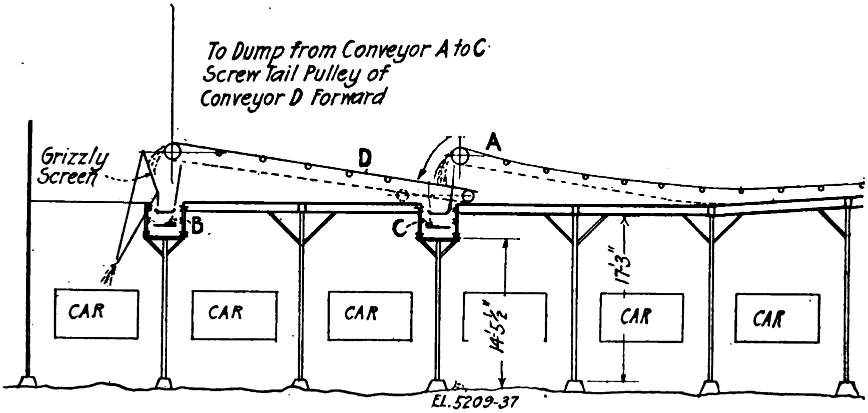


FIG. 14.—PORTION OF SACRAMENTO CONVEYOR SYSTEM.

The inclined belt *A* (Figs. 13 and 14) carries the ore down to the loading tracks, passing over a Blake-Dennison weighing machine on its way.

It is supported in a catenary curve to reduce the height of the trestle. This is now recognized as a mistake, as the maximum inclination is increased and the belt cannot be run taut without lifting it off the rollers when it is run empty, or with a light load.

The line of the incline belt crosses four railway tracks and delivers ore to conveyor *C*, the first of two conveyors parallel with them. The ore is discharged off the end of the belt and falls into a belt feeder similar to that at the shaft. When the ore is to be carried to belt *B*, the end pulley of the short belt *D*, the bearings of which are supported on slides, is screwed back by hand under the stream of ore discharged from *A*. It is then carried by the short conveyor to *B* and dumped into the third

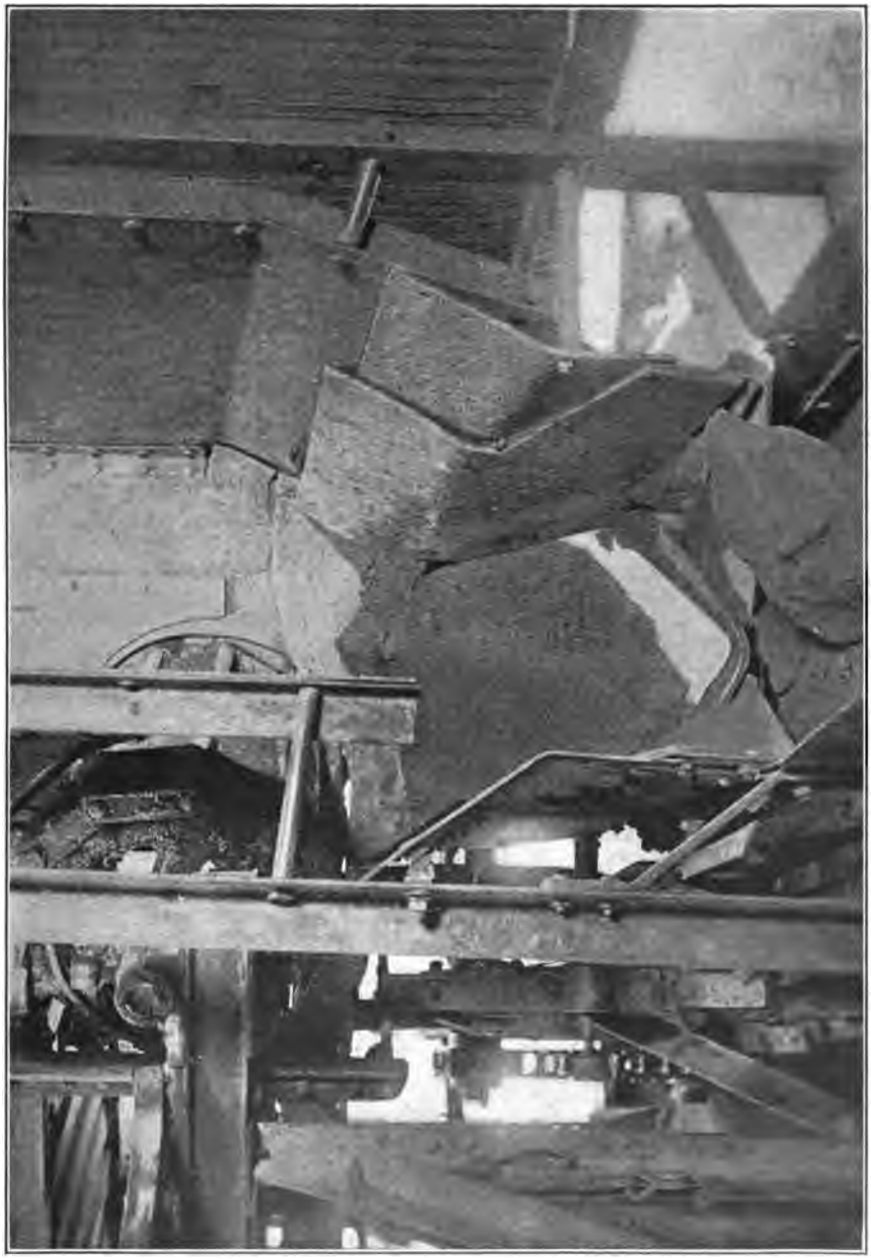


FIG. 15.—BELT FEEDER.

belt feeder. The short belt thus takes the place of a tripper on a continuous belt and reduces the belt cost.

The loading belts *B* and *C* are each located between two tracks, and are long enough to load seven 50-ton cars and a sample car.

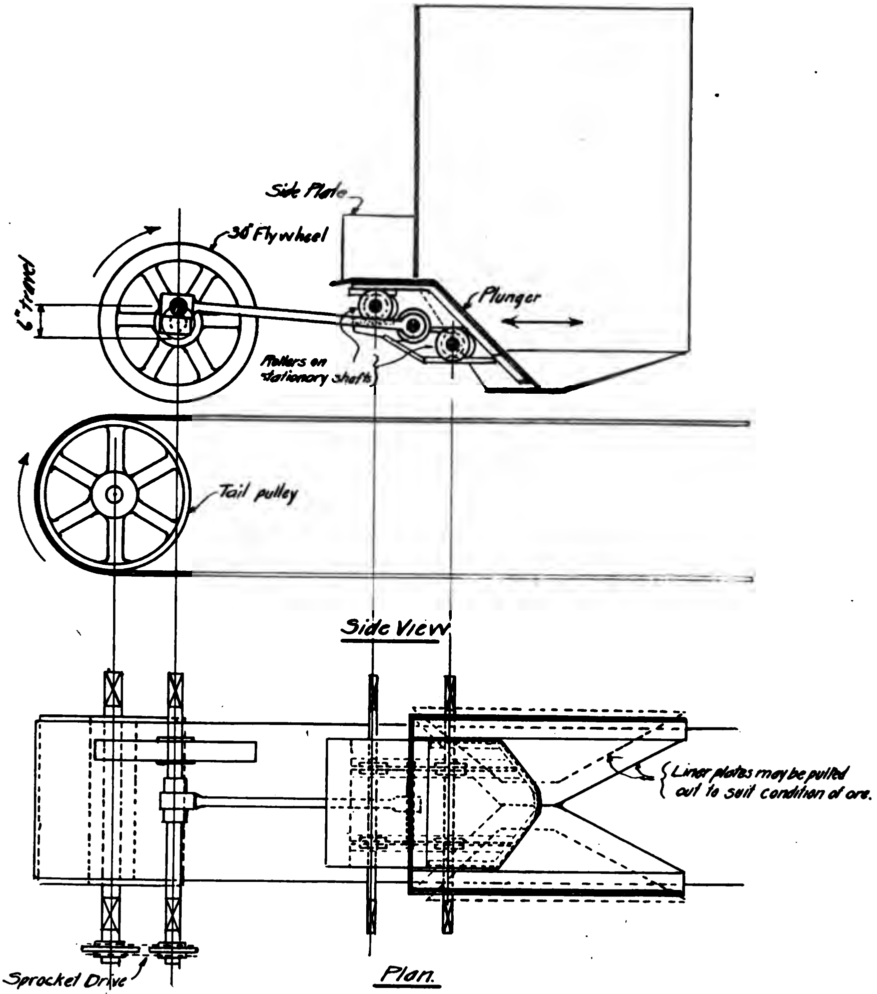

Side Plate
36" Flywheel
6" Throw
Plunger
Rollers on stationary shafts
Tail pulley
Side View
Liner plates may be pulled out to suit condition of ore.
Sprocket Drive
Plan.

FIG. 16.—CONVEYOR PLUNGER FEED.

A tripper (Fig. 17) moves back and forth constantly, depositing layer after layer of ore in the cars below, until they are loaded to capacity as shown by the belt scales. At one end, it travels a few feet over the eighth car and this is taken as a sample of the lot. When the first lot is full, the sample car is switched to the other end of the loading shed and attached to a string of seven cars loading from the other conveyor,

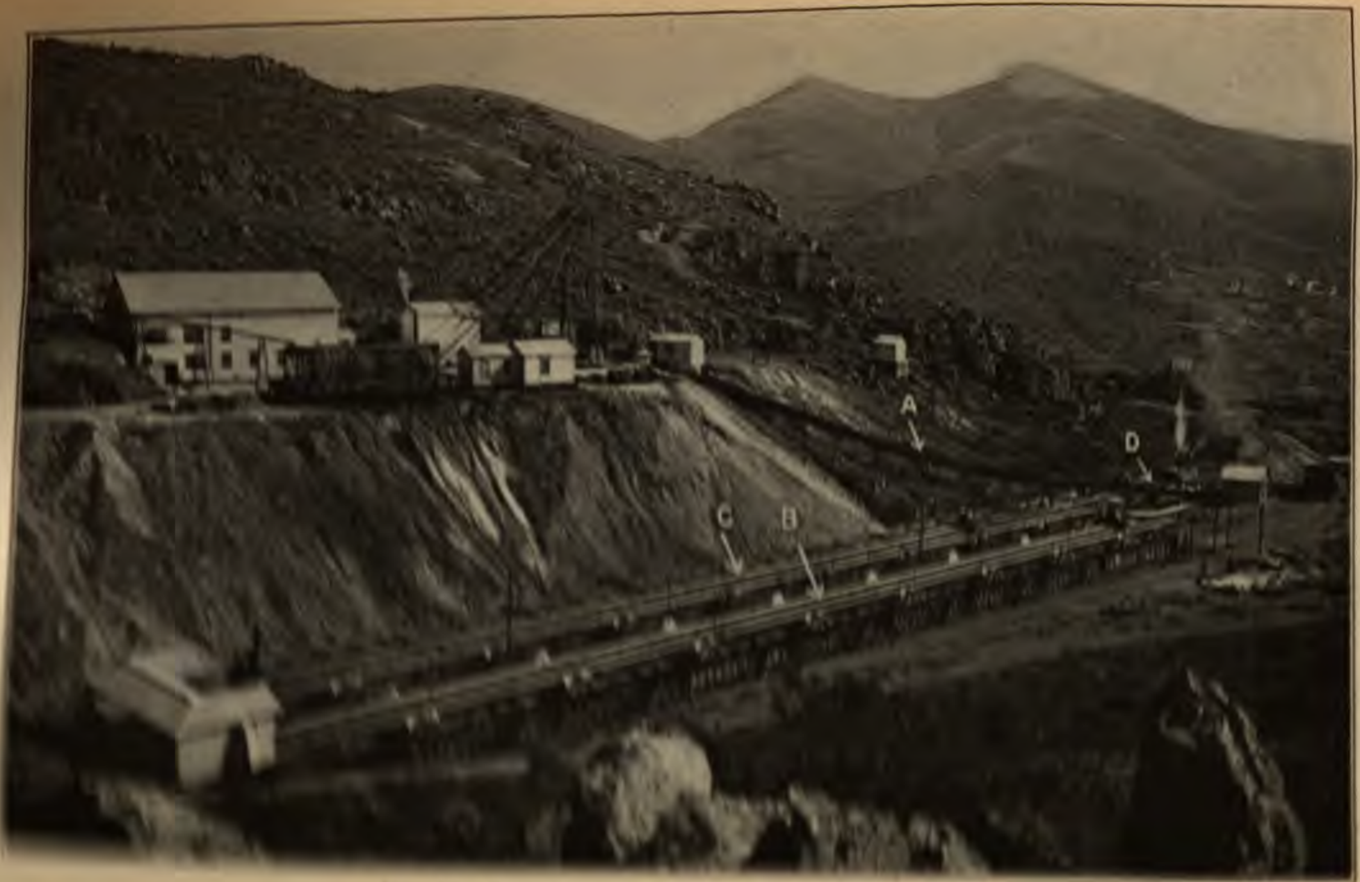


FIG. 18.—SACRAMENTO SHAFT LOADING PLANT IN 1900.

and the sample of another 350 tons is dropped into its opposite end. Its sample therefore represents two lots of ore of 350 tons each.

The belt speeds may be varied from 240 to 360 ft. per minute, but rarely exceeds 300. The tripper speed is from 150 to 190 ft. per minute, but at the maximum speed of the belt would be 225. It travels back and forth from 36 to 40 times in filling a lot, depending on the weight of the ore; thus depositing from 72 to 80 layers of ore in the cars, and taking half as many samples while doing so. Although each lot may be of widely different composition, the ore in each car or in each linear foot of the car is the same as any other in the same lot. At the smeltering plant, the ore is dumped into pits, one lot directly on the other, until a bed of 10,000 tons is made up. It has been found that when loading by steam shovel from one end of this bed, the ore encountered in working through to the other end is sufficiently uniform in composition to secure good metallurgical results. The sampling is also sufficiently accurate.

In its travel, the stream of ore from the tripper crosses a spout 12 in. wide, which catches a sample, splits it, returning one-half to the car, and drops the remainder into a heap on the ground, which is quartered down and assayed for quick returns. It is surprising how closely this extremely rough sample checks with the smelter returns. Although there is considerable variation in the sample of individual lots, the average for a month's run is rarely more than 0.1 per cent.

The first spout tripper was not positive in operation, and a man was needed to ride on it to keep it clear when loading mud. A special tripper was built, in which the ore is discharged upon a short 36-in. belt running at right angles to the main belt. It is driven by a small reversible motor carried on the tripper, which collects its power from a trolley wire, and may load into either of the string of cars spotted below. The spout tripper was built so high, to secure a steep incline for the spout, that it was unsteady and it was impossible to prevent the main belt from running out of line and cutting the edges. The belt tripper is low and steady and the main belt runs more truly and with less wear.

The tripper belts cost 0.266 c. per ton of ore handled. Now that there is a smaller proportion of muddy ore, a low heavy spout tripper might be better. Less attention than before would be needed to keep the spouts clear.

The inclined belt *A* is driven by a 30-h.p. direct-current variable-speed motor at the lower end. The belt travel can be varied between 240 and 360 ft. per minute. The pan conveyor at the shaft is driven from the upper pulley by a sprocket chain, as is the belt feeder also. Belt *D* is also driven from this motor, through a friction clutch. Very little power is required, except to start, about 10½ h.p. with, and 8 h.p. without belt *D*. When loaded and in motion, it will drift so far that it is equipped with a brake in order to stop it quickly when necessary. The two loading belts *B* and *C* are driven by a 30-h.p. variable-speed direct-current motor at the far end, through friction clutches. The tripper belts are driven by 5-h.p. motors.

The trippers are pulled back and forth by ½-in. steel cables driven through reversing clutches, geared to the drive shafts of the belt. Buttons on the rope throw the clutches over at points of reversal, by turning the three-way valve of a compressed-air cylinder, the piston of which acts on the clutch levers. The buttons and points of reversal may be changed quickly, but there is little alteration in the routine and it is rarely necessary. The operation is positive and very satisfactory since powerful clutches have replaced the lighter ones formerly used. The conveyor belt passes idly through the tripper, and there should be less wear than if it supplied the power to propel it.

The control of the whole plant is located on a platform between the loading belts and near the incline. Here are the starting switches for all motors, brake for the incline belt, and levers connected by cables with the clutches at the driving end of the belt conveyors. The belt speed is governed by the speed of hoisting. They are run faster when hoisting from the upper levels. They are stopped when the skips are delayed or stopped for any purpose. The conveyor man is stationed at this point, where he can see all the belts and everything that goes on. One Mexican helper oils up rollers, chains, pulleys, etc. Another is stationed at the shaft to look after the loading end. A mechanic on day shift has been found desirable to attend to current repairs, to re-lace belts, etc. Other men are required from time to time to make more extensive repairs, clean tracks, attend to sampling, etc.

Many changes have been made in the original installation. Belt feeders have been added. The incline belt has been cut in two to eliminate a tripper. The trippers have been rebuilt. The control of the driving mechanism has been centralized. A roof has also been added to protect it from the weather.

The ore is rough, uncrushed, and very hard on belts. The life of belts is gradually increasing, due to improvements in the plant and a better understanding of the conditions.

Recently, it has been desired to screen a certain class of ore for fettling in the reverberatory furnaces, and to supply a coarse sulphide for pyritic smelting. This has been done by raising the discharge end of belt *D* so that the ore falls against a screen above the belt feeder. The fine ore falls into cars directly below, the reject into the belt feeder and makes up a regular lot and sample of coarse screened ore. Sufficient for present needs can be screened out in this manner, but if more were required, more costly changes would be needed.

Increase in Underground Storage Capacity Contemplated

After the changes in the shaft storage bins, no others were necessary to reach the capacity desired or increase the certainty of operation. Alterations have been for purposes of economy or convenience. There have been few delays caused by accident, and none which stopped operations for more than a few hours.

As stated above, the most rapid hoisting in one day was 998 skips, containing 3,467 dry tons of ore, in three shifts of 7½ hours net working time each. There was not enough ore to fill out the third shift at the same rate as the other two. The rate for an hour or two has exceeded this considerably. The maximum rate for any month was in February, 1914, when 65,901 dry tons were hoisted in 58 shifts, or 1,136 per shift. When hoisting at this rate, the shaft bins have not sufficient storage capacity. Time is wasted in changing frequently from level to level to avoid blocking the haulage motors. It is to prevent this that the 260-ton bin on the 400 was put in. Two others are in contemplation, which would increase the storage capacity to 1,350 tons. The hoisting capacity of the shaft would be increased, and there would be fewer delays to electric haulage.

Compressed-Air Hoists for Men and Timber at Most Shafts

The engine for men and timbers at the Sacramento shaft is a compound, two-cylinder engine, made by the Nordberg Manufacturing Co., but drives through Wuest herringbone gears. The cylinders are 18 in. and 30 in. by 42 in. stroke, the gears are 32 in. and 73 in. in diameter, with drums 9 ft. in diameter by 48-in. face, grooved for 1¼-in. rope. The brakes, clutches, etc. are actuated by oil under pressure from the same accumulator which the ore hoist uses, and in a similar manner. The two triplex pumps to supply pressure for the auxiliaries have plungers 2¹³⁄₁₆ in. in diameter by 6 in. stroke, but the oil used when landing several decks is more than it can supply. Three-deck cages in balance are used.

When ore hoisting was transferred to the Sacramento shaft, the Holbrook, Spray and Lowell steam hoists were connected up to drive by compressed air from the central power plant.

The Gardner hoist was at first driven by steam from the power house, delivered through a 5-in. main, 920 ft. long. Upon making tests on the efficiency of hoisting by compressed air, the latter was found to be more economical, and it is now driven by air also.

No changes were made in the engines when using compressed air. The cylinders for brakes, clutches, etc. use cold air. The leakage has been less than when hot air was used and there has been less trouble with packing.

The air for the cylinders is heated to a temperature of 225° to 275° F. The cylinder of the heater is filled with heavy high-test oil, which flashes at about 700° F., and is inclosed in a brick setting like a small boiler. It is heated by a small coke fire under the front part, which is covered by a brick arch to protect the oil from overheating. The gases return to the stack in front through fire tubes in the lower third of the cylinder.

The air makes three passes through the cylinder in small tubes. An open pipe from the dome carries off any products of distillation. The temperature of the oil ranges from 250° F. to 275° F., although it may be carried higher without undue loss of its lighter elements.

The reheater supplies a reservoir of heat for the heavy drain on the system when the engine is starting, and does not over-heat if the engine is idle for a short time.

At the Lowell shaft, the air is heated to an average temperature of 250°. The air consumption there is estimated to be 30,000,000 cu. ft. of free air per month, at a gauge pressure of 92 lb. In the past five months, the consumption of coke has been 3.8 tons per month.

The oil grows slowly thicker by the gradual loss of its volatile constituents. At a temperature of 275° F. it has been used for two years before getting too thick to circulate freely enough to keep the temperature of the whole quantity uniform, and avoid overheating a part of it. The usual life is 9 to 12 months. When too thick for further use, it makes an excellent rope compound. It preserves a rubbery consistency without hardening in cold weather, and does not melt. In fact, it has to be thinned somewhat before the rope takes it well.

Air Stored in Six Cylinders

An air-storage system, composed of six cylinders of 2,500 cu. ft. capacity each, is connected by a 24-in. pipe to a reservoir above. The reservoir holds 4 ft. of water, and is set at such a height that it exerts 93 to 91 lb. pressure on the tanks as it is full or empty. A drop in pressure from 93 lb. to 91 lb. will empty the reservoir and fill the air tanks with water, forcing 15,000 cu. ft. of air at 91 lb. or approximately 100,000 cu. ft. of free air back into the system. This eases the variable hoisting engine load.

Electric Hoist at Czar Shaft

The Czar engine was an old steam hoist for single-deck cages, built by the Union Iron Works, in San Francisco, about 30 years ago. The gear shaft was lifted out of its bearings, and replaced by another, geared to a 112-h.p. alternating-current motor. It has since been replaced by an electric hoist driven by a 225-h.p. alternating-current motor. It hoists a double-deck cage at 800 ft. per minute, but it is usually operated in balance. It is driven through a direct line from the power house, where there is sufficient power in reserve to obviate the necessity of a fly-wheel set or other method of cutting down the peak load. It is started, however, by a General Electric automatic magnetic controller, to prevent too heavy a drag at the time of acceleration.

Brakes are operated by hand, the clutches by air pressure. Power for the latter was added after it was found that the method of throwing them by hand was too slow, when making many changes, as in cage hoisting it is necessary to do.

Several of the shafts are most inconveniently situated for waste disposal. At the Czar, Holbrook, and Lowell, the waste is dumped from the mine cars at the shaft into large cars holding 75 to 100 cu. ft., which are pulled out to the waste pile, by small hoists, and dumped automatically. One of them is pulled up an incline and returns by gravity, and one runs away by gravity and the empty car is pulled back by an engine.

Careful tests of the efficiency of different methods of hoisting were made, the conditions governing the tests being shown in the following paragraphs. Comparative results of the tests are shown in Table IV.

Conditions of Hoisting Tests

Test No. 1

Double-drum engine, reels 7 ft. diameter by 5 ft. face, 1¼-in. rope. Driven by duplex, tandem compound, condensing steam engine, with Corliss valves. Dimensions, 18 and 28 in. by 48 in. Cut-off by governor. Brakes, etc., actuated by hydraulic pressure, condenser and circulation pump by electricity

Hoisting 3½-ton skips in balance.

Average depth of hoisting 825 ft.; maximum depth 1,580 ft.

Boilers isolated for test and feed water weighed.

Steam main 5 in. by 619 ft.

Condensation in main 625 lb. per hour.

Ore weighed on conveyor at shaft, on Blake-Dennison scales.

Work done in changing levels calculated from weight of skip.

Test No. 2

Double-reel flat-rope engine, ropes ½ by 5½ in.

Driven by a 20 by 60 in. simple, duplex, non-condensing engine, with Corliss valves. Cut-off by governor. Brakes, etc. actuated by steam. Boilers close to hoisting engine.

Hoisting triple-deck cages in balance from a maximum depth of 1,600 ft.

Boilers isolated for test and feed water measured.

Ore or waste weighed on scales at collar of shaft.

Work done in changing levels calculated from weight of cages, etc.

Steam for auxiliaries was generated in another boiler, and amounted to 16.1 lb. per shaft horsepower hour, making a total consumption of 111.6 lb.

Test No. 3

Double-reel flat-rope engine, ropes ½ by 5½ in.

Same engine as No. 2, and same hoisting conditions, but engine driven by compressed air.

Air measured by displacement of compressor run especially for this engine. Allowance is made for volumetric efficiency. Transmission losses are practically all leakage, not included, since the proportion chargeable to hoisting depends on the quantity of air passing through the main.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 4

Double-reel flat-rope engine, ropes ½ by 5½ in.

Driven by a 21 by 60 in. simple, duplex, compressed-air engine, with Corliss valves. Cut-off by governor.

Hoisting triple-deck cages in balance from a maximum depth of 1,032 ft.

Air measured by displacement of compressor run especially for this engine, allowance being made for volumetric efficiency and line leakage, which was measured.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 5

Double-reel flat-rope engine, ropes ⅜ by 5½ in.

Engine driven by a 20 by 48 in. simple, duplex, compressed-air engine, with Corliss valves, with cut-off by governor.

Hoisting triple-deck cages in balance from a maximum depth of 943 ft.

Air measured by displacement of compressor run especially for this engine, allowance being made for volumetric efficiency and line leakage, which was measured.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 6

Double-reel flat-rope engine, ropes ⅜ by 4 in.

Engine driven by a 16 by 42 in. simple, duplex, compressed-air engine with Corliss valves; cut-off by governor.

Hoisting double-deck cages in balance from a maximum depth of hoisting 645 ft.

Air measured by displacement of compressor run especially for this engine, allowance being made for volumetric efficiency and line leakage, which was measured.

Air heated by coke heater.

Auxiliaries for brakes, etc., driven by cold air.

Ore or waste weighed on scales at collar of shaft.

Test No. 7

Double-drum geared engine driven by 225-h.p. induction motor. Current three phase, 2,200 v.

Started by automatic magnetic control.

Hoisting double-deck cages in balance from a maximum depth of 417 ft.

Power measured by wattmeters at power house, and includes line losses.

Every fifth car of ore or waste was weighed on scales at the collar of the shaft.

Equivalent pounds of steam and cost of power at power house per shaft horsepower hour are high, due to inefficient turbines at power house and also low load factor.

Test No. 8

Conditions similar to Test No. 7.

TABLE IV.—*Tests on Hoisting Efficiency*

Test Number	1	2	3	4	5	6	7	8
Date of test	4-19-11	4-6-09	2-18-12	2-25-12	3-3-12	3-3-12	6-4-15	6-14-15
Duration of test	8 hr.	7½ hr.	2 hr. 4 min.	1 hr. 16 min.	49 min.	30 min.	6 hr. 30 min.	3 hr. 15 min.
Vacuum, inches	19.8	....	....	....	....	....	....	....
Motive power	Steam	Steam	Air	Air	Air	Air	Electricity	Electricity
Pressure gauge, pounds	137	125	89	89	87	88	....	....
Temperature, °. F.	....	....	205	150	220	200	....	....
Work done, shaft horsepower hours	897.46	335.20	82.5	45.94	22.13	4.75	48.92	33.23
Power consumed per shaft horsepower hour, hoist alone	29.08 lb. steam	95.5 lb. steam	1,489 cu. ft. free air	1,966 cu. ft. free air	1,897 cu. ft. free air	2,322 cu. ft. free air	....	2.47 kw-hr.
Including condenser	31.94 lb. steam	....	....	....	....	....	....	....
Including line loss	34.29 lb. steam	....	....	....	....	....	2.93 kw-hr.	2.6 kw-hr.
Including auxiliaries	34.29 lb. steam	111.6 lb. steam	....	....	....	....	....	....
Including reheating	....	....	1,602 cu. ft. free air	2,177 cu. ft. free air	2,037 cu. ft. free air	2,494 cu. ft. free air	....	....
Equivalent pounds steam per shaft horsepower hour at power house	34.29	105.07	52.87	71.84	67.22	82.30	82.04	72.8
Cost of power per shaft horsepower hour	$0.0137	$0.0399	$0.0288	$0.0392	$0.0366	$0.0448	$0.0465	$0.0416

In Test No. 2, the 111.6 lb. of saturated steam at 125 lb. pressure is reduced to 105.07 lb. in order to make an equitable comparison in efficiency of all tests. In all other tests, the equivalent pounds of steam are based on 150-lb. pressure with 75° F. of superheat, which are the power-house conditions.

In Tests Nos. 3, 4, 5, and 6, the cost of re-heating air appears as an amount of compressed air equal to it in cost.

In Test No. 7, the estimated line loss is 3 per cent.

It is assumed from power-house figures that 33 lb. of steam will compress 1,000 cu. ft. of free air to 93 lb. gauge.

It is also assumed that 28 lb. of steam at the power house generates 1 kw-hr. at the power-house switchboard.

The cost of available steam at power-house is 45 c. per 1,000 lb.

The cost of air at the power-house is 1.8 c. per 1,000 cu. ft. of free air compressed to 93 lb. gauge.

The cost of electricity at the power house is 1.59 c. per effective kilowatt-hour at the power-house switchboard.

The double- and triple-deck cage hoists are at a disadvantage compared to skip hoisting, since two or three lifts of 6 to 8 ft. must be made to land the decks. This only appears as so many feet of hoisting in work done, but it means getting the load in motion, which is the least efficient part of the work, twice or thrice instead of once.

This is exaggerated in the case of the Czar hoist by the shallow depth of hoisting. It will be noted that the decrease in power per shaft horsepower hour in test No. 8 as compared to No. 7 is due to a greater percentage of the material to be hoisted coming from the 400 level. When hoisting from that level, the current consumption per shaft horsepower hour was 2.29 kw.-hr.; from the 200 level it was 2.92 and from the 100 level 4.6 kw.-hr. per shaft horsepower hour. The record of this test is not representative of electric hoisting, and is included as a matter of interest only. Another hoist of the same type will shortly be erected at a 900-ft. shaft, and further tests will be made.

The simple steam hoist is inefficient at best, and particularly so in intermittent service, which does not, however, particularly affect either the compressed-air hoist or an electric hoist without fly-wheel motor generator.

It was a great economy to be able to use the hoisting engines as they stood, even if they were not quite so efficient as those especially designed for air.

TABLE V.—*Ore Haulage by Locomotive*

	1914, Cents per Ton	Four Months, 1915. Cents per Ton
Train labor	6.0	3.5
Loading labor	4.0	4.5
Motor maintenance	2.0	1.1
Car maintenance	2.2	1.3
Trolley maintenance	1.2	0.6
Track maintenance	2.0	1.4
Transfer chutes	2.2	2.8
Power	1.3	1.3
Miscellaneous	1.1	2.2
Foremen and bosses	1.0	0.9
	23.0	19.6

The average distance trammed per car in 1914 was 2,939 ft., returning empty.

Car repairs are rather higher than normal, since the equipment is growing old. Most of the cars have been in service for five years.

The charge for labor indicates that one man takes care of three to four miles of main trolley line, as well as bonding, and a proportion of the transmission line in the shaft. Trolley lines in heavy ground require much attention.

Track maintenance includes renewals and all other repairs. One man cares for 1½ miles of track. Track and trolley extensions are not included. In 1914, this amounted to 5.3 c. per ton.

In 1914, 1.2 c. per ton were spent in transfer-chute construction, and an equal amount in 1915. Much permanent work has been done, and future costs are expected to be lower.

TABLE VI.—*Ore Tramming by Hand or Mule*

	1914	Four Months, 1915
Labor	10.8	9.5
Mules	1.0	1.3
Car repairs	1.6	1.3
Track repairs	3.2	3.3
Miscellaneous	3.3	2.8
Foremen and bosses	0.8	0.8
	20.7	19.0

The cost of hand and mule tramming is so intermingled that it is impossible to separate them. Ore is trammed 194 ft. by hand and 708 ft. by mule. All new light track is charged to this account, whether for new drifts or replacements. In 1914, 47,800 ft. of drift were driven.

TABLE VII.—*Ore Hoisting, Sacramento Shaft*

	1914		Four Months, 1915	
	Total Cost	Cost per Ton	Total Cost	Cost per Ton
Power:				
Steam	$12,632.12	$0.019	$4,216.43	$0.023
Electric	436.72	0.001	....	....
Attendance:				
Engineers	5,694.54	0.009	1,848.25	0.010
Oilers, etc	3,017.44	0.005	931.92	0.005
Skip loaders	7,585.19	0.012	1,844.18	0.010
Cleaning stations	214.30	....	....	....
Miscellaneous	495.48	0.001	800.26	0.004
Hoist repairs	8,702.82	0.013	1,093.63	0.006
Cage repairs	2,549.03	0.004	625.60	0.003
Ore pockets	1,537.01	0.002	981.79	0.005
Hoist buildings	9.05	....	1.51	....
Ropes	3,252.43	0.005	586.39	0.003
Total	46,126.13	0.071	12,929.96	0.069
Tons hoisted	659,102	....	185,179	....

The cost of hoisting is divided into two classes, that of ore hoisting at the Sacramento shaft, and that of handling men, timbers, and waste rock at that and other shafts.

TABLE VIII.—*Conveyor-Plant Costs*

	Cost per Ton 1914	Cost per Ton Four Months, 1915
Power	$0.0037	$0.0027
Attendance	0.0122	0.0129
Belts	0.0026	0.0068
Other supplies	0.0007	0.0008
Repairs	0.0097	0.0078
	0.0289	0.0310
Cost of screening	0.0015	0.0021
	0.0274	0.0289

The rope cost includes that of some defective ropes on which a credit is expected.

Belts are usually charged off more rapidly than they wear out. A better idea of the cost is given by the records of the last belts on each conveyor.

TABLE IX.—*Conveyor-Belt Records*

Belt	*A*	*B*	*C*	*D*	Tripper
Width, inches	30	30	30	30	36
Length, feet	415	700	680	60	16⅔
Tonnage handled	1,166,734	1,070,363	984,130	353,621	51,710
Cost per ton, cents	0.122	0.242	0.288	0.087	0.226
Cost per ton all belts, cents	....	....	....	0.965	....

Cage riders were not customarily employed before the change in the hoisting arrangement. They were found to reduce the danger of shaft accidents.

It will be noted in Table X that the costs at the shafts handling on ore are much more in the aggregate than those at the Sacramento shaft. They are necessary evils, to be held down to the lowest terms.

In 1907, the usual equipment for a producing shaft was: First-motion double-reel engine; double- or triple-deck cages; single-drum man engine; single-deck cage; boiler plant; blacksmith shop; drill- and tool-sharpening shop; timber-framing mill; drill repair shop.

In 1915, the plant is reduced to the first-motion hoist, to which an air heater has been added. The drill- and tool-sharpening shop, framing mill and drill repair shop were not eliminated by the change in hoisting methods, but followed the tendency toward centralization. The expense for surface labor has been reduced in proportion, although it appears principally as a credit to mechanical labor and other surface accounts. The fuel consumption has been approximately cut in two, although the change rooms at each shaft must be heated by independent plants instead of by exhaust steam from the engines.

TABLE X.—*Cost of Handling Men, Timber and Waste Rock*

	Uncle Sam	Southwest	Czar	Holbrook	Spray	Gardner	Lowell	Sacramento Man Engine	Total	Per Ton
Power:										
Steam	......	......	......	......	.....	......	......	$7,634	$7,634	$0.011
Air	$1,100	......	......	$2,667	$628	$6,540	$6,023	......	16,958	0.026
Air reheaters	......	......	......	565	158	911	492	......	2,125	0.003
Electric	......	$744	$1,472	71	.....	747	234	243	3,512	0.004
Labor:										
Engineers	885	4,816	3,799	5,724	1,387	7,358	5,004	4,808	33,779	0.049
Oilers	......	......	......	1,662	548	1,782	1,997	1,666	7,655	0.011
Cagers	739	2,404	4,349	6,071	808	9,233	5,421	2,837	31,861	0.047
Cleaning stations	......	......	329	133	115	376	687	......	1,642	0.002
Miscellaneous	......	......	......	105	.....	......	......	......	105	......
Hoist repairs	886	1,036	2,416	1,475	519	2,283	2,550	1,695	12,860	0.020
Cage repairs	12	250	2,035	850	292	756	1,641	706	6,552	0.010
Rope repairs	90	93	......	......	.....	......	......	......	183	......
Ore pockets	......	......	116	......	.....	......	......	......	116	......
Hoist buildings	......	......	511	......	.....	......	......	......	511	0.001
Rope maintenance	......	......	32	315	1	1,701	927	113	3,272	0.005
	3,712	9,352	5,059	19,638	4,456	31,687	24,976	19,702	128,582	0.189
	Four Months, 1915									
Power:										
Steam	......	......	......	......	.....	......	......	1,985	1,985	0.010
Air	......	......	......	858	.....	1,927	1,626	......	4,411	0.022
Air reheaters	......	......	......	327	.....	229	245	......	800	0.004
Electric	......	380	619	18	.....	312	1	......	1,329	0.007
Labor:										
Engineers	......	766	1,216	1,848	.....	2,209	1,266	1,172	8,478	0.042
Oilers	......	......	......	544	.....	541	654	546	2,284	0.012
Cagers	......	640	1,600	1,621	.....	2,036	1,216	992	8,106	0.040
Cleaning stations	......	......	176	116	.....	43	87	......	422	0.002
Miscellaneous	......	......	......	89	.....	......	......	......	89	.....
Hoist repairs	......	283	427	442	.....	466	699	77	2,394	0.010
Cage repairs	......	53	345	192	.....	855	1,283	457	3,185	0.015
Hoist buildings	......	23	10	......	.....	......	27	......	60	......
Rope maintenance	......	165	104	141	.....	626	613	3	1,653	0.009
		2,310	4,497	6,196	.....	9,244	7,717	5,232	35,196	0.173

As a whole, the costs of tramming and hoisting have not been reduced as much below the costs of 1906 as expected. A reduction in costs due to an improvement in method has been counteracted by the gradually increasing area of operation and necessary equipment. If a comparison might be made with former methods, developed and extended to cover the present work, it would be all that was claimed for it and more.

Although a saving in repairs was counted upon in the original estimates, it was not specified. In 1914, underground repairs cost 51.2 c. per ton and in the first four months of 1915, 40 c. During the last three months of 1905, they cost 59.8 c. for labor alone, and for the first six months of 1906 the cost was 62 c. for labor and supplies.

Individual shafts are not so vitally essential as before. In 1911, the gases from a mine fire rendered the Lowell shaft impassable to men between the 800 level and the surface, for about five months. For two months, timber and supplies were lowered and waste was hoisted through the gas, but since no repairs could be made, a guide finally broke loose and jammed one of the cages. During this period, workmen were lowered through neighboring shafts and ore was hoisted as usual. Both production and costs for the year were normal.

The writer wishes to express his indebtedness to members of the mine department staff for assistance in collecting data used in this paper; W. Saben, Chief Clerk, Charles Legrand, Consulting Mechanical Engineer, R. E. Cameron, a member of his staff, and George Mieyr, Master Mechanic.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Pacific Coast Iron Situation

The Iron Ores of California and Possibilities of Smelting

BY CHARLES COLCOCK JONES,* B. S., LOS ANGELES, CAL.

(San Francisco Meeting, September, 1915)

In any discussion of this very large subject we are confronted at the outset with so many obstacles that at best only a fragmentary and rather disconnected presentation can be made of it, and my hope is that building from the few facts I have to offer a more complete knowledge may be secured.

In the whole intermountain and Pacific Coast region, west of Pueblo, Col., and from Canada to Mexico, there is no production of pig iron, and steel production is confined to the manufacture of a small amount of open-hearth steel from scrap. When we consider the area, the resources, and the several centers of population, this condition seems anomalous, even in the face of every counter argument as to cheap water transportation, or other means of transport from the Atlantic Seaboard or from the iron and steel centers of the Middle West.

This comatose condition, if such a term may be applied, is certainly not due to a lack of all the essential materials and factors for success in iron and steel smelting, but is rather, I think, due largely to the general fact that it is only within a comparatively few years that the centers of population on the Pacific Coast have attained a size that would entitle them to the position of distributing or freight centers, or that the "back country" has been enough built up to make it sufficiently interesting to the railroads to foster trade from the local centers to the interior points rather than seek to supply such local centers and the interior by a transcontinental haul.

ORE SUPPLY

Mr. Edwin C. Eckel, in his monumental work on *Iron Ores*, has but little to say concerning the Pacific Coast deposits. He mentions (p. 269) those of Minaret, Madua County, and the Eagle Mountains, Riverside County, Cal., "perhaps 50 to 100 million tons each;" the deposits in Shasta County, worked for the electric furnace on Pitt River; and

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* Consulting Engineer.

"ores in Oregon and Washington" which "have been slightly developed;" and while he admits that (by reason of unfavorable conditions, as to labor and fuel, for the manufacture of iron) there has been comparatively little prospecting for iron ores in the Pacific States, he thinks that certain geological types of iron-ore deposits, which have been highly productive in the East, are probably wanting here—for instance, sedimentary beds like the Clinton, and brown-ore deposits like those of the Appalachian region. And in his final summary of American iron-ore resources (p. 349) he cites Hayes' estimate of 126,000,000 tons of available ore "for the Rocky Mountain and Pacific States;" pronounces it too low, since Utah alone contains more; and finally puts down the Western States for a minimum of 300,000,000 and a possible maximum of 700,000,000 tons. His estimate for the whole United States being 5,200,000,000 minimum and 7,500,000,000 maximum, the proportion assigned to the great area between the Rocky Mountains and the Pacific is notably small.

Mr. Eckel does not cite, in his Bibliography of California deposits, *Bulletin* 503, *U. S. Geological Survey*, by E. C. Harder, on Iron Deposits of the Eagle Mountains, published in 1912. In the light of the Harder report on the Eagle Mountains and other data, I think the Eckel estimate of from 300,000,000 to 700,000,000 tons can be safely applied to the Pacific States alone, excluding the tonnages of Utah and the other Rocky Mountain States; and, from our present knowledge, most of it is in Southern California.

At the annual meeting of the American Mining Congress held in Los Angeles in 1910, the writer, in an address on the subject of Iron Ores of the Southwest, called attention to the large available deposits of high-grade iron ore in southern California and adjacent territory, and previously in the *Engineering and Mining Journal* of April 17, 1909, had described technically the iron deposits in the Providence Mountains of San Bernardino County, California. The U. S. Geological Survey published in 1908, *Bulletin* 338 on The Iron Ores of the Iron Springs District, Southern Utah," and in 1912, *Bulletin* 503, Iron Deposits of the Eagle Mountains, California. Descriptions of various other iron deposits are scattered through technical publications.

Epitomizing the subject in 1910, I came to the conclusion that there are 200,000,000 tons of available, high-grade iron ore and double that amount of probable ore in southern California. In addition to this an equal amount can be counted on from Lower California and Mexico, which will naturally seek an outlet to the northward.

This estimate does not include the Utah deposits.

In 1908 and 1909, the late Edward H. Harriman acquired the Eagle Mountain iron deposits, in Riverside County, 50 miles north of Mecca, a station on the Southern Pacific Railroad, 140 miles east of Los Angeles, with the publicly avowed intention of establishing a steel works at Los Angeles. The U. S. Geological Survey states as a "moderate estimate" that there are 60,000,000 tons of iron ore in this deposit. Later, what is known as the Cave Canyon iron deposit, 200 miles east of Los Angeles on the Salt Lake Railway, containing 20,000,000 tons, was bought by the same interests. The untimely death of Mr. Harriman in 1909 put a damper on further development, and his vision is yet to be realized.

Near Kelso, 236 miles east of Los Angeles, is the Vulcan iron deposit, containing from 6,000,000 to 10,000,000 tons of high-grade, soft hematite ore of Bessemer grade.

In the Kingston Mountains at the California-Nevada line, north of the Salt Lake Railway are other large deposits.

The Colorado Fuel & Iron Co. owns a deposit of 12,000,000 tons 10 miles west of Silver Lake on the Tonopah and Tidewater Railway.

There are large amounts on the Santa Fe Railway in San Bernardino County south of Newberry, north of Amboy and south of Blythe Junction, and numerous other deposits are known in the desert region. The three most practically available deposits, viz.: the Eagle Mountain, Cave Canyon, and Providence Mountain deposits, represent an aggregate tonnage of 125,000,000 tons, and have the characteristics necessary to the best possible furnace mixture for economical smelting. These ores run from 4 to 10 per cent. higher than standard Lake Superior iron ores. The bulk of the tonnage is within the Bessemer limit in phosphorus and, lacking moisture, there is no cost of drying before transportation, or freight cost on 10 per cent. of moisture as with Lake ores. *Bulletin* 503, on Iron-Ore Deposits of Eagle Mountain, California is authority for these statements. The average analysis of Providence Mountain iron ore is:

Iron, 64.82; phosphorus, 0.044; silica, 3.04; manganese, 0.278; alumina, 0.568; magnesia, 0.201; lime, 0.444; sulphur, 0.059; titanium, none; volatile, 1.85 per cent.

CONSUMPTION

According to one of the leading buyers of iron and steel, the following is a conservative estimate of the present approximate yearly consumption on the Pacific Coast:

	Tons
Pig iron	100,000
Reinforcing steel for concrete buildings	30,000
Sheets	136,000
Plates	217,000
Corrugated	15,000
Bar	175,000
Structural	250,000
	923,000

Probably the aggregate is more nearly double this estimate. Complete figures would include the large amounts of cast-iron water pipe, oil-well pipe, railroad steel and other steel products not listed above.

COSTS OF PRODUCTION

At some future time there will certainly be one iron-smelting center in Southern California; and there may possibly be a second at San Francisco and a third at Seattle, each having its own distinct province from which to draw its supplies of raw materials; but at the present time Southern California seems to have the advantage in the combination of ore, fuel, labor and climatic conditions for economical production.

The statement that conditions justify the establishment of at least one plant, based on the consumption of iron and steel products on the Pacific Coast will be borne out, if it can be shown that these products can be manufactured locally at a price to compete with the supply from other smelting centers.

In the following discussion some of the figures relating to Pittsburgh are used as given in a recent publication by the Merchants' Association of New York on the iron and steel industry, supplemented by personal knowledge, and figures from *Mineral Resources of the United States*, 1913, published by the U. S. Geological Survey, and also figures from the report of the Commissioner of Corporations on the Steel Industry, 1911, and costs of making pig iron from testimony before the Ways and Means Committee of the House of Representatives in 1906.

Pittsburgh is the center of the iron and steel industry in the United States and any costs on the Pacific Coast that can be shown to approximate Pittsburgh costs will demonstrate not only the feasibility of the industry, but the certainty of profits.

The difference in mining and transportation costs between Lake Superior ores and California ores is due to a number of reasons, principal among which are the following: The Lake Superior ores are obtained either by expensive underground mining or by open-cut work in which large amounts of overburden have to be removed at considerable cost; and, on account of climatic conditions, the shipping season is limited to seven months of the year. Transportation is a combination of rail, water and rail, entailing several handlings of the ore.

In the case of California ores, we have large surface exposures of orebodies, capable of being quarried and loaded on cars by steam shovel, and handled thence direct to the furnaces. The mining cost of California iron ores will be on the same low basis as that of the porphyry copper ores, or from 20 to 30c. per ton. The figure of 85c. in the following estimate is ample to cover both royalty, or sinking fund, and mining. Climatic conditions are such that work can be carried on every day in the year with the exception possibly of a few stormy days; and California ores have the added advantage of a higher content in iron and a lower amount of gangue to be fluxed, resulting in greater efficiency, or tonnage product of a furnace and decreased cost of flux, per ton of iron produced, all of which factors materially lower the operating cost per ton of pig iron produced.

TABLE I.—*Comparison of Costs of Iron Ore and Transportation at Pittsburgh and Los Angeles*

	Pittsburgh Lake Superior Ores	Los Angeles California Ores. Assuming Max. Haul 260 Miles at 0.005c.
Rail and Lake rate to Lower Lake Ports	$1.15	
Rail freight rate, Lower Lake Ports to Furnace location.	0.88	
Rail freight rate, Eagle Mountains, Providence Mountains, and San Bernardino County ores.		$1.30
Total transportation, costs per ton	$2.03	$1.30
Average mining cost per ton	$2.40	$0.85
Total transportation and mining cost per ton	$4.43	$2.15
Percentage iron yield per ton ore, non-Bessemer grade, per cent.	51.5	57.00
Percentage iron yield per ton ore, Bessemer grade, per cent.	55.0	64.00
Total transportation and mining cost per ton pig iron, 95 units iron, non-Bessemer grade.	$8.17	$3.56
Total transportation and mining cost per ton pig iron, Bessemer grade.	$8.05	$3.19

Comparing the Bessemer ores alone, as the bulk of the California ore is of that quality, the advantage of Los Angeles over Pittsburgh, in cost of iron ore per ton of pig iron is $4.86.

The average mining cost for Lake Superior ores is taken from *Mineral Resources of the United States*, 1913, U. S. Geological Survey, and is exclusive of any transportation costs.

Coke Supply

Of equal importance to the iron ore is a supply of suitable coal for coking. There are probably good coking coals in Alaska and Washington that could be placed in the Los Angeles market at a reasonable price; and there are large coal fields in southern Utah at present without railway connection, undoubtedly containing coking coals; but at the present time the coking coal of Carbon County, Utah, is the most available by reason of rail connection. The Carbon County, Utah, coal, at Sunnyside in the Book Cliffs coal field, compares favorably in coking and chemical qualities with some of the Eastern coals and a comparison of costs on that basis will be made.

An average analysis of Sunnyside coal and coke is as follows:

	Coal. Per Cent.	Coke. Per Cent.
Moisture	1.27	0.13
Volatile matter	39.13	0.83
Fixed carbon	54.62	88.96
Ash	4.98	10.07
	100.00	99.99

The metallurgical coke for the Utah and Montana copper smelters is supplied from the Sunnyside field and its good qualities are thoroughly proven. A description of the Book Cliffs Coal Field was given by Robert S. Lewis.[1]

In the report of the Commissioner of Corporations on Steel Industry, 1911, the cost of coke at Pittsburgh has been shown to be $2.97 per ton of pig iron, and from that cost it is estimated that the cost of byproduct coke, provided a profitable market for the byproducts could be secured at Pittsburgh, would be $1.94 per ton of pig iron. The above figure of $2.97 is borne out by the current quotations on Connellsville coke, May 1, 1915. As compared with the cost of byproduct coke at Pittsburgh, the following costs are conservative for Los Angeles, where a ready market could be found for the byproducts and where a number of chemical industries could follow the establishment of a byproduct coke plant, utilizing the coal tar, ammonia and gas.

TABLE II.—*Cost of Coke at Los Angeles or Southern California Points*

Mining cost 1.65 tons Utah coal at 90c. per ton		$1.48
Transportation 1.65 tons at $4.25, 850 miles at 0.5c		7.01
Total cost coking and byproduct operation per ton coke		1.40
		9.89
Credit from sale of byproducts per ton coke, 8,000 cu. ft. gas at 15c.	$1.20	
14 gal. tar at 4c	0.56	
24 lb. sulphate ammonia at 3½c	0.84	2.60
Net cost of coke per ton		$7.29
Total cost of coke per ton of pig iron (1.10 tons per ton pig)		$8.02
Cost byproduct coke Pittsburgh, minimum		1.94
Maximum advantage of Pittsburgh in cost of coke per ton of pig iron.		6.08
Total cost of coke per ton of pig iron (1.10 tons per ton pig)	8.02	
Or at average cost of coke	2.97	
Average advantage of Pittsburgh in cost of coke per ton of pig iron	$5.05	

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[1] *Bulletin* No. 91, July, 1914, pp. 1729 to 1749.

The building of the Alaska Railroad into the Matanuska coal field by the government will open that coal to Southern California; and, if the claims regarding its coking qualities are substantiated, it will afford a supply of coking coal cheaper than from any other section.

In Table II the coking question has been treated from the standpoint of making byproduct coke at tidewater in order to improve the coke, secure the byproducts, and have a source of supply of power in the waste gases for use in a steel plant independent of any one coal field; but the coke supply can also be considered from the standpoint of beehive-oven coke made in the coal fields and transported to tidewater, if the cost of such coke can be shown to be less than that of byproduct coke.

I should observe that my estimates are made on a rational freight rate of 0.5c. a ton a mile. The present published rates on coal and coke from the interior are higher than that, being still based on "all the traffic will bear;" and, since the amount handled is small, no attempt has been made to secure a lower rate. That 0.5c. a ton a mile is a reasonable maximum is shown by the fact that that rate is in existence for phosphate rock and other material, of a like or higher value than coal, from this intermountain region to tidewater. I mention this matter because it is one of the "scarecrow arguments" that have to be met. When the traffic justifies it, the rate can be secured, through the usual channels.

Limestone Supply

Available to every railroad entering Los Angeles, at distances of from 20 to 150 miles, are ample deposits of limestone suitable for blast-furnace flux. This rock can be quarried and brought to tidewater at as low a price as that used in Pittsburgh, and from many of the deposits for a less cost. Even at a greater cost per ton of limestone, the cost per ton of pig iron would still be as low as the Pittsburgh cost on account of the greater purity of the iron ores and consequent smaller amount of lime used for flux. In my estimate of final costs, the Pittsburgh figure will be applied as fully covering the limestone cost.

Application of Above Figures and Comparison of Cost of Pig Iron at Pittsburgh with the Estimated Cost at Los Angeles

In 1906, before the Ways and Means Committee of the House of Representatives, Judge Gary testified that for all the furnaces of the United States Steel Corporation the total cost of making a ton of pig iron, based on the market prices of iron ores, coke, limestone, cinder and scale, and including maintenance and depreciation of furnaces, was $15.30 per ton of pig iron, divided as follows:

TABLE III.—*Cost of Pig Iron at Pittsburgh*

Iron ore per ton $4.70, per ton pig iron	$8.62
Coke per ton $3.93, per ton pig iron	4.15
Limestone $1.06, per ton pig iron	0.49
Scrap, cinder and scale	0.27
Labor and maintenance	1.37
Depreciation of furnaces	0.40
	$15.30

From these figures he said the following deductions should be made for net profits:

On ores in pig iron	$2.04	
On coke in pig iron	0.60	
On transportation	1.07	$3.71
Actual cost of pig iron		$11.59

Assuming other costs than the ore, coke and flux to be the same at Los Angeles and Pittsburgh, we have the following as a conservative statement of the estimated cost of a ton of pig iron at Los Angeles, or tidewater.

TABLE IV.—*Estimated Cost of Pig Iron at Los Angeles*

From Table I—Total transportation and mining cost per ton pig iron, Bessemer grade ore	$3.19
From Table II—Total cost of coke per ton pig iron	8.02
Limestone	0.49
Scrap, cinder and scale	0.27
Labor and maintenance at furnaces	1.37
Depreciation of furnaces	0.40
	$13.74
In *Mineral Resources*, 1913, Part 1, p. 321, the government gives as the average price per ton of pig iron at the Pennsylvania blast furnaces for 1913	$15.36
And for Alabama furnaces	12.08

Accepting for the purposes of this discussion the figures given by Mr. Gary of $11.59 as the net cost of pig iron for all the furnaces of the United States Steel Corporation in the year 1906, and deducting it from the estimated cost at Los Angeles of $13.74, we have a difference of $2.15 per ton in favor of Pittsburgh, but the railroad freight rate on pig iron from Pittsburgh makes the net cost of pig iron at least $20.95 f.o.b. Los Angeles and selling price $23.00 to $25.00 per ton.

In regard to freight rates on pig iron and steel products to Pacific Coast points, the published rate from Chicago and Birmingham is $9 per ton, and there is no through rate from the Pittsburgh district, and from Pittsburgh to Chicago takes a local rate of $3.15, making the total freight from that point $12.15 per ton to California terminal points.

Steel products have a uniform freight rate of $16 per ton, Pittsburgh to Coast points. In my calculations an average freight rate of $9 has been used, being the rate from Chicago or Birmingham to Coast points.

It should be noted particularly that our comparison of costs is with the costs of a corporation owning mines, railroads, vessels and smelting plants and its costs are necessarily lower than the merchant-furnaces in Pennsylvania which buy ore and coke, and a deduction from Tables I and II more nearly states the relative costs.

From Table II, advantage of Pittsburgh in cost of coke, per ton of pig iron.....	$6.08
From Table I, advantage of Los Angeles in cost of ore, per ton of pig iron......	$4.86
Net advantage of Pittsburgh in cost of ore and coke per ton of pig iron........	$1.22

If, however, in all fairness to Southern California, the figure $5.05, the *average advantage,* instead of $6.08, the *maximum advantage* of Pittsburgh in cost of coke per ton of pig iron is used, as shown in Table II, we have 19c. only as the *net advantage* of Pittsburgh in cost of ore and coke per ton of pig iron. *In other words, iron ore is the costly item and coke the cheap item at Pittsburgh, while in California the costly item is coke, and iron ore is the cheap item, with the difference in the cost of the two materials about equalized.*

From Table I, the cost per unit of iron, Lake Superior iron ore, Bessemer grade at Pittsburgh, is 8c.; the cost of California iron ore, per unit of iron, at Los Angeles is 3.33c.

Markets and the Cost of Reaching Them

At the present time the entire Pacific Coast is dependent upon the East and South for its supply of pig iron and steel, with the exception of a small amount of open-hearth steel produced from scrap at San Francisco and Los Angeles. The imports of foreign iron and steel are a negligible quantity and are never likely to be of any importance on account of the cost of transportation.

Therefore, as our principal market, we have this Pacific Coast province, comprising the States of California, Oregon and Washington, which can only be reached by the eastern manufacturer after paying a minimum transcontinental freight of $9 per ton, or a minimum water freight from Atlantic ports of $6 per ton.

Los Angeles is the great lumber market of the Pacific Coast, and most of the vessels in the trade are without cargo on the return trip and would welcome the opportunity to carry iron and steel products to San Francisco, Portland, Seattle and Vancouver at low water-rates. The markets of South America are open on more than equal terms; for tonnage through the Panama Canal from the Atlantic seaboard to the west coast of Mexico and Central and South America must pay the toll of $1.20 per ton.

Simply stated, pig iron can be made in the regular blast furnace way at Los Angeles for less than $14. No Pennsylvania pig iron, either by transcontinental freight or by rail to New York and via Panama Canal, can be put into this market under $19 per ton. Alabama pig might be put in at a slightly less cost, but its quality would not place it in the same class as pig iron made from California ores.

Geology

The principal Southern California iron deposits are of the contact-replacement type formed at or near the contact of an intrusive igneous mass with sedimentary strata, usually limestones. Their occurrence has been fully discussed in Iron Ores of the Iron Springs District, Southern Utah, by C. K. Leith and E. C. Harder, *Bulletin* 338, *U. S. Geological Survey*, 1908; Iron Ore Deposits of the Eagle Mountains, California, by E. C. Harder, *Bulletin* 503, *U. S. Geological Survey* (1912); and An Iron-ore Deposit in the California Desert Region, by C. C. Jones, *Engineering and Mining Journal* (April 17, 1909); and to these accounts I will add only the following note, bearing somewhat on a recent discussion in the Institute.

The Providence Mountains of Eastern San Bernardino County present the greatest uplift and cross-section of strata found between the Grand Canyon of the Colorado in Northern Arizona and the Sierra Madre range in Western San Bernardino County, and afford an opportunity to study a section approximately 4,000 ft. thick, from the underlying granitic and schistose rocks up through a sedimentary series predominantly composed of limestones with subordinate beds of shale and quartzite. This has been brought to the attention of the U. S. Geological Survey; and I believe, an extended study of the region will fill several gaps in our knowledge of the geology of this part of the desert region.

The general strike of the sedimentary beds is northeast-southwest with a slight dip to the east. The central core of the mountain, exposed in canyon from base to top, and forming the highest peaks, is a core or plug of rhyolite, a mile or more in diameter. At the southwest end of the mountain the sedimentary beds, lying horizontal or dipping slightly to the east, and retaining this position undisturbed, have been cut by a large quartz-monzonite mass, the contact between the monzonite and the limestones being practically vertical. I have indicated on the accompanying sketch map the general conditions.

The specific fact I wish to bring out for the attention of those interested in the discussion on The Origin of the Garnet Zones and Associated Ore Deposits,[2] by Waldemar Lindgren, and Recrystallization of Limestone at Igneous Contacts,[3] by C. K. Leith, is the different alterations sustained by the same series of limestones in adjacent areas in contact with the same igneous intrusive rock mass.

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[2] Trans., xlviii, 201 to 208 (1914).

In the sketch map, I have indicated conditions 9 miles southeast of Kelso. *R* is the central core of this part of the Providence Mountains, composed of rhyolite, and detached small rhyolite exposures are shown at several places in the limestones marked *R*. *XY* is the line of contact between the sedimentary series and the quartz-monzonite mass.

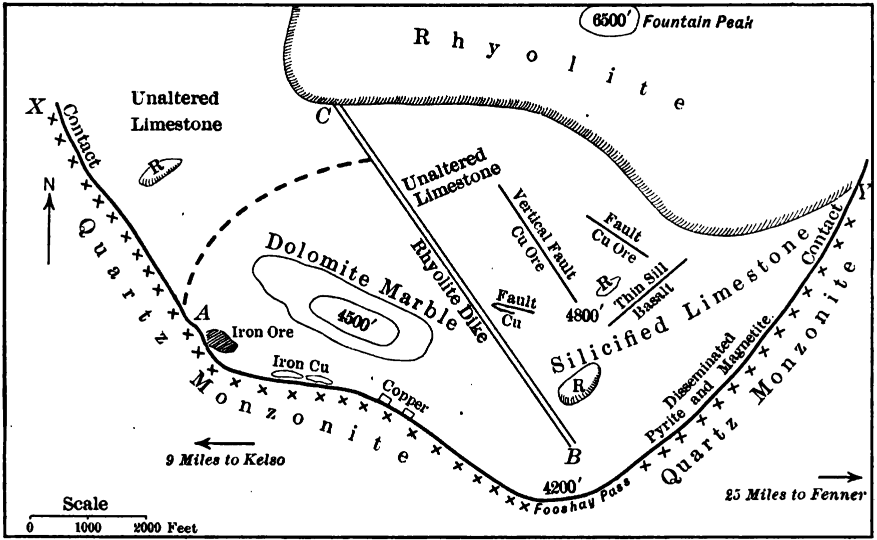

Fig. 1.—Sketch Map Showing Geological Conditions in Eastern San Bernardino County, Cal.

The contact of the monzonite, cutting sharply the practically horizontal beds of limestone, runs for a distance of about 4 miles. *BC* is a vertical dike of rhyolite porphyry, 100 ft. thick, an offshoot from the central core. The point marked *A* seems to have been the center of activity, resulting in a replacement of the limestone by iron, over an area of four acres, with smaller orebodies along the contact to the east and with a great dissemination of grains and irregular replacements of iron ore in the limestones. The original bedding planes of the limestone are still preserved in the beds of iron ore; and in the area *ABC* the limestones have largely lost their bedded structure and a mass of crystalline dolomite or dolomitic marble has resulted. A fuller description of the geologic features is given in my article in the *Engineering and Mining Journal* cited above. The end of the dike *BC* is not seen in actual contact with the monzonite at *B*. To the west of dike *BC* is an area of dolomite, as indicated, of the following analysis:

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[3] Trans., xlviii, 209 to 215 (1914).

	Per Cent.
Insoluble siliceous	0.43
Aluminum and iron oxides	0.24
Calcium carbonate	63.08
Magnesium carbonate	35.97
	99.72

To the east of dike *BC* is an area of limestones between the rhyolite and monzonite, a portion of which is seemingly unaltered, especially just east of point *C*, where, in contrast with the dolomite west of the dike, the blue unaltered limestones preserve perfectly their bedded structure, and the larger portion of the area, especially at and for several hundred feet north of the contact, is largely silicified. An analysis of a typical sample shows:

	Per Cent.
Silica	48.0
Lime	25.8
Iron oxide	4.3
Alumina	12.9
Carbonic acid	5.0
Undetermined	4.0
	100.0

Much analytical work is needed to determine the composition of the original unaltered limestone. While I am not prepared to discuss thoroughly the phenomena involved, my impression is that the original limestones were calcitic, and that ascending heated waters, following the intrusion of the monzonite, decomposed its ferro-magnesian members, deposited the iron as a replacement of the limestone near the contact, and dolomitized the limestones between the contact and the dike *BC*. But the reason of the different action east of the dike *BC* I leave to those better qualified to determine. In the east area are a number of vertical faults, some of great length and some short, nearly all of them showing a slight mineralization, chiefly of copper minerals in calcite veins, the gold and silver contents being negligible. Unfortunately the work done on these veins has not been productive; otherwise a fuller study would, perhaps, have been made of the conditions I have indicated above. The only working mine in the immediate region is situated 9 miles northeast of Fooshay Pass on the same contact, modified by some pegmatite dikes, at what was formerly known as the Bonanza King mine, where several million dollars worth of silver ores were extracted from irregular bodies in the limestones.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Furbero Oil Field, Mexico

BY E. DEGOLYER*, A. B. NORMAN, OKLA.

(San Francisco Meeting, September, 1915)

I. INTRODUCTION

THE Furbero oil field lies in the Gulf coastal plain of Mexico between the important ports of Tampico and Vera Cruz and about one-third of the distance between the two points from the latter (Lat. 20° 22′ N., 97° 30′ W. Gren.). It is in the State of Vera Cruz, about 14 miles west by south of the village of Papantla in the canton of the same name, and is the most southern of the developed oil fields of the Tampico-Tuxpam region being a distance of 50 miles south by east from the Potrero del Llano field and 42 miles in the same direction from the Alamo field. The field has been developed by the Oil Fields of Mexico Co. and is in the midst of a large block of lands owned or leased by them. Communication is through the port of Tuxpam with which it is connected by a narrow-gauge railway and 6-in. pipe line some 52 miles in length.

This field is of more than ordinary interest since it represents a type of oil deposit hitherto unknown. So far as I am aware, no other commercially valuable deposit of oil is known to occur under conditions of accumulation similar to those found here. The igneous nature of the so-called 'sand' encountered in drilling was not recognized until a number of wells had been drilled; and it was not until recently that the geological relations of the sill or laccolith, which gives rise to the accumulation of oil, were known and the so-called 'anticline' was known to be, in part at least, the result of igneous intrusion.

Geological surveys in this general region were made in 1900–1901 by W. H. Dalton and in 1907–1909 by P. C. A. Stewart, F. W. Moon, Leonard V. Dalton, and Dr. Max Muhlberg. More recent surveys have been made, however, under the direction of the author, by E. L. Ickes, B. C. Belt, B. Hartley, and Harve Loomis. Notes of the earlier examinations were not available with the exception of the report by Dr. Muhlberg who apparently first recognized the true nature of the Furbero intrusion. This paper is indebted to his work and that of the more recent surveys for much of the detail regarding stratigraphy and to Messrs. Percy Furber and Arthur C. Payne, President and General manager of the Oil Fields of Mexico Co. for permission to publish, and for much of the information regarding the field. The detailed structure of the intrusion as herein set forth is the result of a study of the well logs and samples by the author.

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*Chief Geologist, Cia. Mexicana El Aguila, S. A., Cuban Oil Co.

II. HISTORY AND DEVELOPMENT

This general region, including Furbero, is one of the earliest known oil regions of Mexico. Indian tradition says that this part of the coast was settled during the seventh or eighth century; the Sabaneta topographic basin, in which Furbero lies, being then known as Choloa. That the existence of oil seepages was probably known to the early inhabitants, is indicated by the fact that the locality now known as Furbero was formerly called "Cougas" (later corrupted to "Cubas"), a Totonaco word, meaning a species of black wax or oil. Moreover, Spanish writers immediately following the conquest note "chapapote"[1] as among the

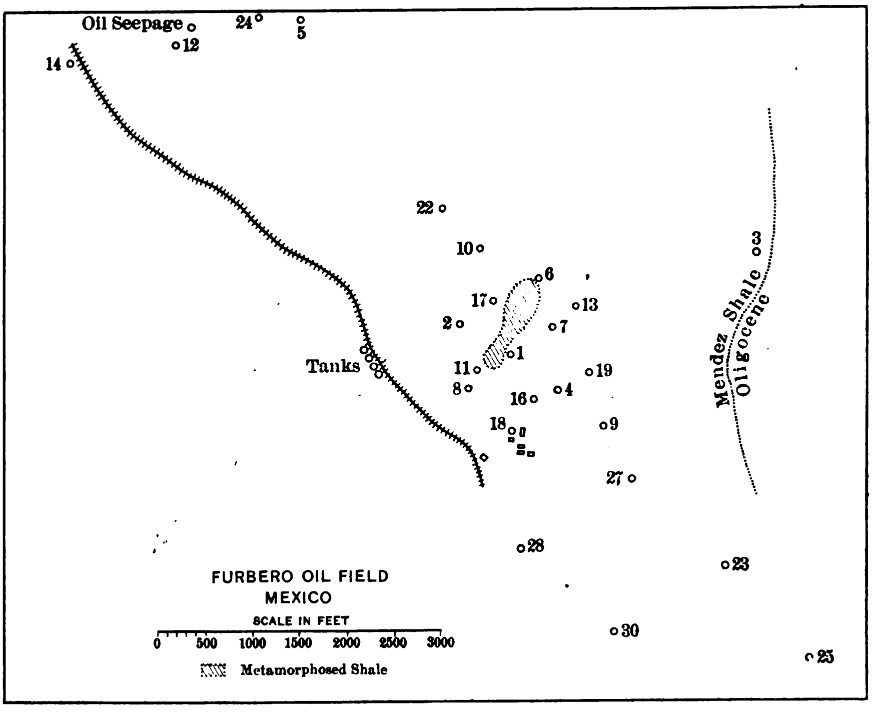

FIG. 1.—MAP OF FURBERO OIL FIELD, SHOWING DEVELOPMENT.

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[1] "Chapapote" is the term by which asphalt, petroleum residue, and even oil itself are known in Mexico at the present day. It is derived from the Aztec words "tzauc," meaning paste or cement and "popochili," meaning perfume and probably refers to two of the early uses of the material. Robelo, Cecilio G.: *Diccionario de Seudoaztequismas*, p. 536 (Cuernavaca, 1906).

articles for sale in the great market of Mexico City. Old workings consisting of a shaft walled up with rock were found near one of the principal seepages by the earliest known white explorer.

In 1868, a certain Dr. Autrey, who had been carrying on the practice of his profession at Papantla for several years, discovered the larger seepages, and the knowledge of their existence and location being made public, a company (the Cia. Explotadora de Petroleo del Golfo Mexicano) was formed at Mexico City in 1869, for the purpose of exploiting the deposits. Machinery was brought into the region and a 3-in. bore-hole was sunk to a depth of 125 ft. near one of the largest seepages. No production resulting, a tunnel was driven 50 ft. into the hill near the old Indian workings, and a moderate supply of oil (reported as about 4 or 5 bbl. per day) was secured. The project was soon abandoned, however, and about 1870 the properties, including abandoned machinery, were denounced by Autrey. One of the old stills was set up in Papantla and the crude oil was packed mule-back to that point where some 4,000 gal. of kerosene were refined and sold locally.[2]

In 1883, the present mining law, vesting the ownership of petroleum rights in the ownership of the surface of the land, was enacted, and the property thus passed out of Autrey's possession. In 1892, El Cougas and 2,500 hectares of land were leased to an American oilman, this being the first oil lease of record in Mexico.[3]

From 1895 to 1900, numerous leases were made in this general region, and in 1901 the Mexican Petroleum and Liquid Fuel Co., Ltd. was formed and took them over. This company was founded with English capital, among the interested persons being Cecil Rhodes of South African fame. During the years 1901 and 1902, drilling was actively carried on, 24 wells being drilled, two of which attained a depth greater than 1,500 ft. Operations being unsuccessful from a commercial standpoint, the region was abandoned. In 1903, the Oil Fields of Mexico Co., the present owner of the Furbero field, was organized. Lands and leases over a great area were secured. Machinery was brought into the region via the Tecolutla River and wagon roads; and in 1904 drilling was commenced in earnest. In 1907, the first well "came in"; and in April, 1910, the pipe line and railroad to the port of Tuxpam having been completed, commercial deliveries of oil began. Since that time drilling and oil production have been almost continuous. Thirty-two wells have been drilled by this company in the general region, 24 of them in the Furbero field proper.

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[2] This early history is taken from a copy of a letter dated Mar. 2, 1892, from Adolpho Autrey, M. D., to John F. Dowling, City of Mexico.

[3] Cuestiones Petroleras de Actualidad, Mexico, 1912; Lic. Edward Schuster, p. 1.

III. TOPOGRAPHY

The region in which the Furbero field occurs is topographically a great basin shut in on the north, west, and south by the great lava-capped mesas which lie at the foot of the central Mexican plateau (the remnants of a flow which once covered the entire region), and on the north and east by Tertiary hills. The major axis of this basin runs northwest and southeast and the drainage is controlled by two master streams: (1) The Cazones River, which emptying into the Gulf of Mexico about 40 miles northwest of Furbero, drains the northern part of the basin, including the Furbero field proper, through its tributary the Arroyo Coachepa and secondary streams; and (2) the Tecoluta River, which empties into the Gulf at a point 30 miles east of Furbero, draining the southern part of the basin. The water-sheds of these streams are separated by a very low divide. The Furbero field proper, as developed hitherto, has an area of little more than 100 acres. It lies in the northeastern part of the basin on the slope from the Tertiary hills to Arroyo Coachepa on the west. The most striking features of the immediate topography are the sharp twin hills in the midst of the field, which resemble very much the small volcanic plugs or necks so commonly features of the topography of the oil regions north of the Tuxpam River. The general elevation is from 600 to 650 ft. above sea level.

IV. GEOLOGY

This region is covered, for the most part, with dense tropical jungle and a mantle of soil resulting from centuries of vegetable decay; and as a consequence there are but few rock exposures of geological value. These are commonly found in the banks of creeks or arroyos. No exposures large enough to admit the measurement of detailed sections of the various formations are found. We are therefore forced to depend mainly upon the soil and topographic expression, verified by occasional exposures, in mapping the surface geology, and upon well records and general regional observations in studying the details of the various formations.

In considering the Furbero field proper, we are directly concerned with the Tertiary marine sediments and the igneous rocks only; but a description of the Cretaceous limestones is here given for the purpose of defining the relation of the Furbero field to other fields of the region.

1. *Cretaceous*

Tamasopo Limestone.—The oldest known rocks of this general region consist of a series of limestones, known from their outcrop in the Tamasopo Canyon, State of San Luis Potosi, as the Tamasopo limestone. This

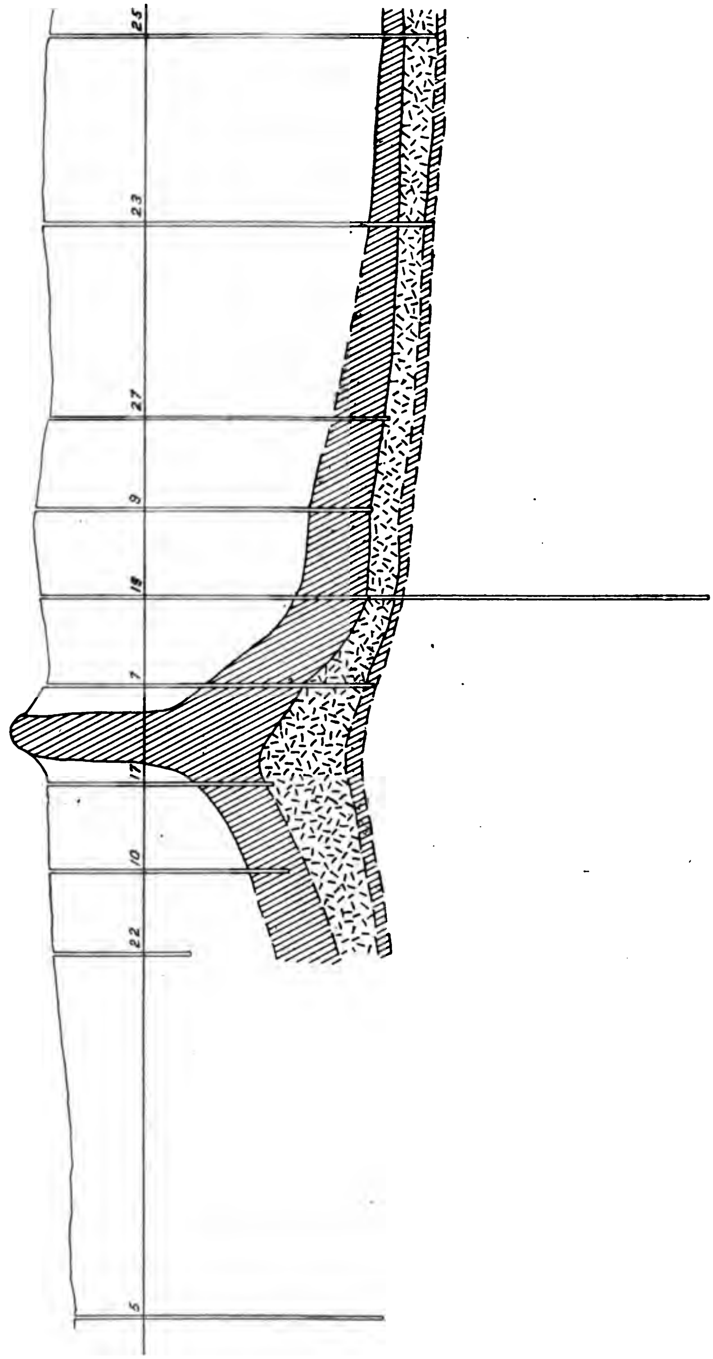

FIG. 2.—GEOLOGICAL SECTION OF THE FURBERO OIL FIELD.

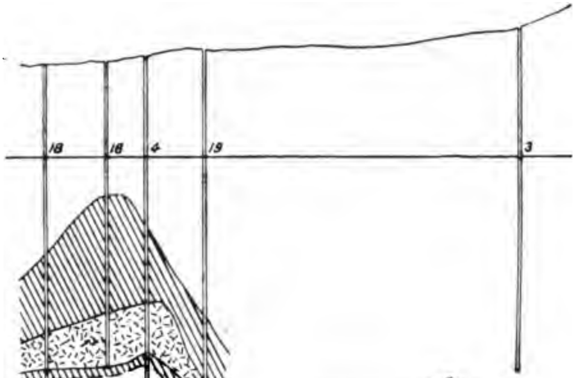

FIG. 3.

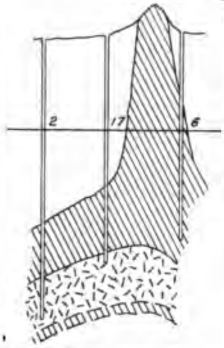

FIG. 4.

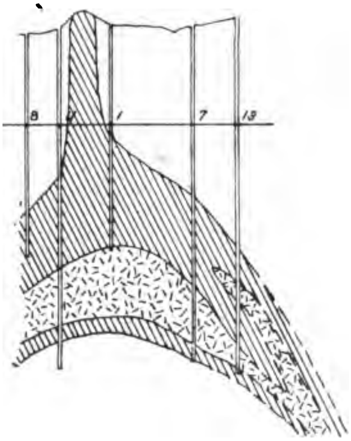

FIG. 5.

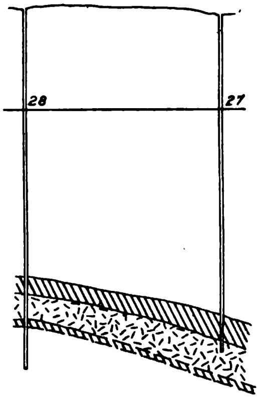

FIG. 6.

FIGS. 3 TO 6.—GEOLOGICAL SECTIONS OF THE FURBERO OIL FIELD.

formation consists of hard, gray, pure, compact, porcelain-like limestones bedded in layers less than a foot thick, and is characterized, in its upper part, by the occurrence of an abundance of black to dark gray and green chert nodules, interbedded with the limestones. Economically, the formation is of greatest importance because of its oil-bearing horizons. The uppermost members of the limestone, which are massively bedded in the northern Vera Cruz and Valles regions, are somewhat porous and contain great solution caverns. These cavernous limestones are the reservoir rocks of highly productive fields of Potrero del Llano, Juan Cassiana, Dos Bocas, Los Naranjos, and Alamo, which are characterized by immense and prolific gushers (daily capacity 50,000 to 100,000 bbl.). The formation is generally regarded as the parent oil-bearing formation of northern Mexico, from which the oil at present found in other formations has come by migration.

The main mass of the outcrop of the formation in this region is in the Sierra Madres, the front range of which passes points respectively 28 miles west and 16 miles south of the Furbero field. The formation has not yet been encountered by wells drilled in this region, but it is very probable that the oil of this field originated in the Tamasopo and was forced upward by hydraulic and gas pressures, along a channel formed by the broken and metamorphosed zone of contact of igneous and sedimentary rocks, or through faults and fissures, to the reservoir from which it is now obtained.

The thickness of the formation in this region is unknown, but probably is from 6,000 to 8,000 ft. The age of the formation is lower Cretaceous, evidently the equivalent of the Comanche of Texas.

2. *Tertiary Eocene*

San Felipe Beds.—Overlying the Tamasopo limestone and resting unconformably(?) upon it, is a series of alternating, impure, thin-bedded limestones and gray, red, and green shales and marls known as the San Felipe beds. The entire formation is somewhat sandy and contains, locally, beds of tuff of variegated colors which contain decomposed mica and are finely porous. It is also reported to contain conglomerate though well records do not show such a rock. The economic importance of the formation in this region has not yet been determined. It has been encountered by two of the wells drilled and oil and gas have been produced from it, but not in great quantities. In the fields of the Panuco River valley near Tampico, it contains the oil-producing horizons. With the exception of one or two doubtful inliers, the outcrop of the formation in this region is confined to a narrow strip adjoining the outcrop of the Tamasopo limestone in the mountain front. The thickness of the formation varies from 600 to 1,000 ft.; it is probably more nearly the latter in this region. The exact age of the formation is unknown but it is apparently

Tertiary if one may judge from the few fossils which have been secured from drill cuttings. If such is true, it is of lower Eocene age. The formation grades imperceptibly into the overlying shales series, the limestones gradually becoming more argillaceous and impure and grading finally into hard shale and, in turn, into soft shale. It is doubtful if the upper 100 to 200 ft. of the formation can be differentiated from the overlying shales in well logs.

Mendez Shale.—Grading from the underlying San Felipe beds, is a thick series of gray to green shales, marls, and clays, containing rarely thin shaly sandstones and limestones and red shales, called the Mendez shale, because of its outcrop near Mendez station, just west of Tampico. Economically, it is generally of no importance; but in the Furbero field, part of its ordinarily impervious shales have been baked and metamorphosed into a hard, brown to black, porous shale and it forms, together with the crystalline igneous rock the oil-reservoir rock of the field. Apparently much of the metamorphism has been due to the action of ascending thermal waters associated with the igneous rock after its intrusion; for the pipe of metamorphosed shales which rises to the surface and outcrops in the twin hills in the central part of the field cannot have been the direct result of the heat of the igneous rock. The igneous rock of the intrusion has also undergone alteration since crystallization.

This formation outcrops for the most part over the entire floor of the Sabaneta basin. It is the surface rock of the Furbero field proper, extending from the Oligocene hills on the east to the lava flows at the foot of the hills of the Sierra Madre on the west. The thickness of this formation at Furbero is approximately 4,000 ft. No fossils have been found in this region; but a collection made at Alazan some 50 miles to the north has been identified by Dr. Dall as lower Eocene.

3. *Tertiary: Oligocene*

Overlying the Mendez shales is a thick series of sandstones, shales, impure fossiliferous limestones and occasional conglomerates of Oligocene age. The various strata making up the formation are lenticular and grade laterally into each other. Near the front of the Sierra Madre occur beds of shale so thick that their outcrops are hardly distinguishable from those of the Mendez shales. The formation is not known to be of economic importance in this region. The local outcrop occurs over an area east of a N-S line just east of well No. 3.

4. *Igneous Rocks*

The igneous rocks of the region consist of basalts, dolerites, basalt-gabbros, and various products of volcanic activity, such as volcanic sands, ash, etc. The surface outcrops consist almost altogether of lava-capped mesas, the remnants left by the erosion of a great lava flow which once covered most of the region. Several big dikes are found near Reparo and many porphyritic basalt dikes cut the Oligocene rocks near Buena Vista. A patch of volcanic ash covers the surface near Palma Sola. The lava flows evidently originated at some point in the Sierra Madre, probably in the vicinity of Necaxa. The flow varies in thickness from a few feet to 500 ft., and it has a uniform inclination of several degrees to the northeast, an inclination due evidently to the topography of the pre-existing land surface. The nearest remnants of this flow to the Furbero field are the lava-capped mesas of El Brinco, 6 miles to the northwest, Mecatepec, 9 miles north, and a mesa near Tecuantepec River, 12 miles east of south. There are no evidences of deformation since these flows, which occurred after the deposition and folding of the entire series of marine sediments.

Of greater interest and economic importance is the occurrence of a laccolith or extremely thick sill, the formation of which, together with associated phenomena, has given rise to the accumulation of oil known as the Furbero pool. This intrusion does not reach the surface. The only indication of its existence is the outcrop of metamorphosed shale already mentioned. It has been well explored, some 16 wells having been drilled into it, five other wells into the overlying metamorphosed shales, and six wells, encountering no evidence of intrusion, in the immediate vicinity. The intrusion evidently commenced in the form of a sill following closely the bedding planes, the sedimentary rocks having already been folded into an anticline. The intrusion thickened near the crest of the anticline, lifting the overlying beds and further accentuating the already existing fold. Small amounts of oil are produced from the igneous rock in several of the wells. As has been observed, the gabbro seems to have been greatly altered and is quite porous in certain localities. The greatest thickness of igneous rock shown by the drill is in well No. 7 where 440 ft. were drilled through. A careful construction of the probable form of the intrusion, however, suggests a maximum thickness of from 600 to 650 ft. near wells 17 and 7. Minimum thicknesses of 193 ft. and 170 ft. have been encountered in wells 19 and 28 respectively, though the actual thickness of the intrusion is probably little more than 100 ft. at well 19; the greater apparent thickness being due to the fact that the intrusion is dipping steeply where encountered by the bore, which therefore does not show a true section. The highest point reached by the intrusion, as shown by well records, is in well No. 17 where the top of the intrusion was reached at a depth of 1,490 ft. or 891 ft. below sea level. The lowest point yet reached is in well 25 at a depth of 2,665 ft. or 1,984 ft. below sea level. The general form is indicated by the accompanying sections. Two sills of igneous rock separated by some 300 ft. of metamorphosed shale were passed through by well No. 13. The area known to be underlain by the igneous rock as determined by wells which have been drilled up to the present time (encountering either the igneous rock itself or the metamorphosed shale) is 26 acres; but the actual area underlain is known to be greater than this, probably several times as large.

5. *Structure*

The general structure of this region, as of the Gulf coastal plain for some distance northward, is that of an eastward-dipping monocline; the oldest rocks—the Cretaceous limestones—outcropping in the steeply folded and faulted ranges of the Sierra Madre and passing under the Tertiary rocks of the coastal plain, and the Tertiaries outcropping generally in the order of their succession, the youngest beds fringing the coast. The beds of the monocline are gently folded, the complexity and steepness of the folding and faulting being greatest in the Sierra Madre, and decreasing with distance from it. Folding took place in Oligocene or post-Oligocene time and afterward the entire region was subject to igneous activity. As has been noted, it was almost covered by an immense lava flow, the remnants of which cover hundreds of square miles at the present time. The sedimentary rocks were also intruded to some degree; but the surface evidences of intrusion are not very extensive.

The general structural strike of this entire region is northwest-southeast, the axes of the minor folds of the Tertiary rocks of the coastal plain being parallel to those of the steeply folded Cretaceous rocks of the Sierra Madre. One of these minor folds, the Furbero anticline, having been complicated by the intrusion of igneous rocks and the metamorphism of previously impervious beds, affords the structure in which the oil of the Furbero pool has accumulated. It may be held by some that the folding of the sedimentary rocks has been due to the intrusion. That they may have occurred contemporaneously is possible; but that the intrusion represents an incident of the more general igneous activity which took place in the Tampico-Tuxpam region after the folding of the sedimentary rocks, seems more probable. That the intrusion followed generally preexisting structure seems probable. The major axis of the Furbero anticline and the major axis of the intrusion are the same and parallel to the axis of the other folds in the general region. The floor of the intrusion is of a shape generally anticlinal and is evidently influenced in form by conditions existing at the time of the intrusion. Had the strata been level when the igneous rock was intruded, one would have expected to find the laccolith with a flat floor, the arching of the beds being confined to those above the plane of intrusion. The dip of the upper surface of the laccolith, though in the same direction as that of the floor, is generally steeper; but the exact effect of the intrusion on the position of the outcropping strata cannot be determined because of scarcity of surface exposures. It seems probable that the intrusion ascended from below in the form of a dike, and encountering a bedding plane of little resistance which was dipping steeply (west flank, sections *B* and *C*) was diverted into that plane, and took the form of a sill. The splitting of the intrusion (well 13) probably indicates the point of this change of form.

V. Details of Deposit

As will be noted from the accompanying map, the laccolith has been intensively explored, several of the wells being only 80 to 100 ft. apart and most of them being within 200 ft. of other wells, so that several have been found to interfere with each other in actual exploitation. Wells more recently drilled in the exploration of the southern part of the laccolith have been located from 800 to 1,000 ft. apart.

This field is structurally a somewhat complicated anticline, and the occurrence of oil may be classed as conforming to the structural theory; yet the determining factor in the accumulation of oil has been the providing of pore space for a reservoir by the metamorphism, accompanying the igneous intrusion, of the under- and overlying shale, which were previously impervious. Of consequence also has been the subsequent alteration of both shales and igneous rock; but, as this was probably an indirect result of the igneous activity, it need not call for separate consideration. Normally, the unaltered sediments were probably quite similar to the unaltered shales into which they grade, both the altered and unaltered shales being of the Mendez formation, a series of blue and gray, medium-soft, fine-grained shales, more or less calcareous in places, and (when not metamorphosed) of fairly constant lithological character throughout. The metamorphosed shales seem to be of two general types: A fairly hard, black shale, differing but little from the unaltered gray shales into which it grades; and a hard, brown, porous, silicified shale, which usually occurs immediately above the igneous rock and grades into the black shales. The zone of metamorphosed shales envelopes the igneous rock which it both overlies and underlies, and is distinctly thicker over the intrusion than under it. Above the apex of the intrusion the metamorphosed zone continues in the form of a pipe or chimney to the surface, where its outcrop forms two sharp hills, resembling in contour the volcanic plugs of the region north of the Tuxpam River. This extension of the metamorphosed zone is probably due to ascending gases or thermal waters, and not directly to the heat of the intruded masses. As is seen on the sections, the upper metamorphosed zone including both the black and the brown shales, varies from 100 to 500 ft., and the underlying zone from 50 to 150 ft., in thickness.

The igneous rock is a dolerite or gabbro which has undergone, in places a somewhat extensive alteration. Drill cuttings of the rock consist for the most part of gray sand much resembling a true quartz sand. This sand, together with the position and structure of the igneous rock, which somewhat resembles that of a thick lens of folded sedimentary rock, was quite conducive to concealment of the true igneous nature of the intrusion; and the geology was first interpreted as that of a simple fold in sedimentary rocks. Pieces of igneous rock several centimeters in diameter have been recovered from the cuttings. A sample from the intrusion penetrated by well No. 13 at 2,400 ft., has a porosity of 6 per cent., as determined by experiment. A sample from 1,835 ft. in well No. 16 is apparently a coarsely crystalline dolerite and is quite porous. A characteristic log of a well in this field as partly determined from well samples, is as follows:

Furbero Well No. 11	Thickness, Feet	Depth, Feet
Yellow clay (surface)	29	29
Blue and gray shales with oily seams, between 400 and 900 ft. (unaltered).	1,221	1,250
Black and brown shales of variable hardness (metamorphosed) with good shows of oil at 1,650 and 1,655 ft.	415	1,665
Gray sand (igneous crystalline; good show of oil at 1,650 ft. contact)	363	2,028
Black shales (metamorphosed)	122	2,150
Gray shales (unaltered)	200	2,350
Small production of oil at the upper contact zone of igneous and sedimentary rocks.		

VI. Occurrence of the Oil

The oil in this field occurs in commercial quantities both in the igneous and metamorphosed rocks. Well No. 27, the largest well in the field, though it was already producing while in the upper metamorphosed shales, "came in" with an initial production of almost 1,000 bbl. per day upon being drilled 30 ft. into the igneous rocks. It has produced more than 200,000 bbl. of oil, most of which has apparently come from the igneous-rock reservoir. Well No. 28 also showed an initial production of 150 bbl. per day from a depth of 150 ft. into the igneous rock. Well No. 9 has produced approximately 150,000 bbl. from the upper metamorphosed shales; and most of the production of the remaining wells has come from the metamorphosed shales, the zone of contact with the igneous rock being particularly productive. Salt water has been encountered immediately under the oil in several of the wells.

As has been observed, the greatly varying porosity of the rock, rather than the structure, is believed to have been the determining factor in the accumulation of the oil of this deposit. The most productive wells of the field have been Nos. 9 and 27, and they lie to the east of the center of the area of greatest production. The crest of the laccolith or dome lies altogether outside the most productive area, though some oil has been encountered in every well drilled to a depth at which it might reasonably be expected. In my opinion, the intrusion would have been as effective in any form which it might have assumed, provided that it did not outcrop.

From a knowledge of this deposit and of the Mexican fields in general, I believe that the oil of this deposit originated in the Cretaceous limestone, and has been forced upward to its present position along the metamorphosed zone of contact of the sedimentary and igneous rocks.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Manufacture and Tests of Silica Brick for the Byproduct Coke Oven

BY KENNETH SEAVER,* S. B., PITTSBURGH, PA.

(San Francisco Meeting, September, 1915)

INTRODUCTION

It is rather gratifying that in the field of the manufacture of refractories, in which, as a whole, foreign practice has surpassed us, the United States stands pre-eminent in the making of silica brick. In some instances European manufacturers have refused point-blank to credit the possibility of making to such specifications as are common with us, until the actual completed shapes were shipped for inspection. But both the manufacturer and consumer in foreign lands are now frankly learning from us, and benefiting by our advanced practice in this field, as is evidenced by the orders placed here and the efforts put forth abroad to come up to the standard of American-made material.

In no metallurgical operation has the employment of silica brick spread with such leaps and bounds as in the building of plants for the carbonization of coal, particularly in the byproduct coke oven. Today, in practically all types of oven, the structure above the floor level (except the facings) is of silica, while in the recuperative and in some regenerative types, its use is extended to that portion of the oven below the floor as well.

By reason of the fact that there are a number of classes of refractories of higher silica content than the usual so-called fire-clay brick, there is sometimes a misunderstanding as to the class of material referred to as "silica" brick. We here refer only to a brick having a silica content of 94 per cent. or more and made usually from quartzite rock with a small percentage of lime as a binder. The term "silica" brick is sometimes rather loosely applied to brick from highly siliceous clays, or to a quartzite refractory of mingled fire-clay and quartz rock; but such products are not considered in this discussion.

MATERIALS

The rock from which practically all high-grade silica brick are made is a true quartzite or metamorphosed sandstone, variously known as ganister, quartz, quartzite, silica rock, etc.—the largest fields, in the order of their relative importance being those of Pennsylvania, Wisconsin, Alabama and Colorado.

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Pennsylvania Quartzite

Pennsylvania manufactures probably from 75 to 85 per cent. of all the silica brick made in the United States; nearly all are made from the quartzite found in the Medina and Oneida formation. While this formation extends across the State of Pennsylvania reaching into New York State on the north, and Maryland, Virginia and still farther toward the South, the quarries are practically all in Huntingdon and Blair Counties, Pennsylvania. It is in this restricted area that the quartzite is at its best. Toward the north and toward the south it becomes softer, is no longer a true quartzite, or is too much impregnated with iron. The rock is also comparatively easily accessible in the counties mentioned, enormous quantities lying loose upon the surface. The particular occurrence of this loose rock has here been brought about by the Juniata River and its tributaries, as they have cut across the formation, wearing away the softer formations beneath, and leaving the Medina overhanging and exposed. In the course of time it has been weathered and broken up by the elements into the bodies of loose rock or "floes," as they are called, which may cover an area of a thousand acres or more. The depth usually varies from three or four to as much as 20 ft. Smaller floes formed in the same way may be found along the flanks of the mountains; but they nowhere compare in size with those formed where the measures have been cut transversely.

This rock is sometimes quarried out of the solid measures, but by far the largest amount used is that taken from the bodies of loose rock, as the latter is not only cheaper to work but is of more uniform quality. The white Medina formation is not composed entirely of quartzite, but is made up principally of sandstone with intermingled strata of quartzite. Thus where the formation has been broken up, the softer sandstones unsuitable for the manufacture of silica brick have tended to weather away and disintegrate, thus leaving the larger portion of the floes consisting of quartzite. When sandstone is mixed with the quartzite in the floes it is readily detected by its appearance, as it tends to become much more rounded and worn in weathering than does the quartzite, which maintains its sharp angular edges.

The individual pieces in these floes range from those of fist-size to boulders weighing many tons. Probably 50 per cent. of the rock occurs in pieces weighing more than 500 lb. The largest floes are found at the gorges of the Juniata River near the towns of Mount Union, Mapleton, Point View, Lewistown, etc.

It is interesting to note that the Oriskany sandstone, extensively used as glass sand, is found in close proximity to the Medina and that although exceedingly pure in its composition, its softness renders it unsuitable for high-grade silica brick. It consists of a mass of minute quartz grains loosely held together, and often so friable that it can be easily worn away by the hands. On the other hand, the Medina quartzite consists of quartz crystals cemented in a mixture of the same composition and of equal hardness with the crystals. The difference in the two types of rock when ground to sufficient fineness for the manufacture of brick is obvious. The harder stone gives splintery, angular fragments; the other, rounded grains. Experience shows that a hard rock like the Medina is required for a physically strong silica brick.

The white Medina which occurs in Southern Pennsylvania near the town of Hyndman is there prominent and of a pure variety, but by no means as hard as the quartzite of Huntingdon County.

A representative analysis of the Pennsylvania quartzite is as follows:

	Per Cent.
Silica (SiO_2)	97.80
Alumina (Al_2O_3)	0.90
Ferric oxide (Fe_2O_3)	0.85
Lime (CaO)	0.10
Magnesia (MgO)	0.15
Alkalies	0.40
	100.00

Wisconsin Quartzite

This rock, much used in the manufacture of silica brick in the vicinity of Chicago, is known as the Baraboo quartzite from the town of that name where these measures appear, and near which the quarries are located. The rock occurs in high cliffs throughout this region and is especially prominent around Devil's Lake. Here glacial action has excavated a broad deep valley in the quartzite formation bounded on each side by cliffs several hundred feet in height. Their bases are covered with the loose rock split from the parent ledges. While the fragments are much larger as a whole than the weathered Medina of Pennsylvania, the quality by no means compares with the latter. Practically all the stone used in brick making is here quarried from the solid measures.

The marked peculiarity of this rock, as compared with the Pennsylvania quartzite, is its reddish color, which ranges from a light pink through a dark purplish red almost to black. It is often as red as the red Medina of Pennsylvania, a sandstone wholly unsuited for brick making. While the coloring of the Baraboo quartzite is due to iron, it is yet an exceedingly pure rock. The quartzite often becomes lighter in color as the quarry works in from the face of the cliff, but it is the exception to find white rock.

This Baraboo quartzite, estimated as some 4,000 to 5,000 ft. in thickness, is a metamorphosed sandstone, the sedimentary origin of which is indicated by its parallel bedding planes, by ripple marks, etc. Layers of quartzite are often separated by beds of quartz schist varying from a few inches to several feet in thickness. This schistose structure is due to the admixture during deposition of clayey material, which in the subsequent folding and bending of the rock has developed a slablike or schistose structure.

The United States Geological Survey explains the formation of the quartzite as due to the enlargement of the original quartz grains and to the deposition of interstitial quartz. It is the latter that carries the iron oxides, which give the characteristic color to the rock. Regardless of its color, the formation throughout shows the physical characteristics of hardness and strength, as well as the splintered and angular fracture in crushing, identical with the Medina quartzite of Pennsylvania. A typical analysis follows:

	Per Cent.
Silica (SiO_2)	97.15
Alumina (Al_2O_3)	1.00
Ferric oxide (Fe_2O_3)	1.05
Lime (CaO)	0.10
Magnesia (MgO)	0.25
Alkalies	0.10
	99.65

Alabama Quartzite

The material used in silica brick manufactured in Birmingham, Ala. is found in the eastern part of that State. It is bluish white, and as hard and crystalline as the Wisconsin quartzite; average analyses show it to be equally pure. It occurs in lenses of varying size, in a sandstone formation. It is probably a quartzite formed by a leaching out of impurities from the parent rock followed by a crystal growth and additional deposit of silica from circulating waters. A typical analysis follows:

	Per Cent.
Silica (SiO_2)	97.70
Alumina (Al_2O_3)	0.96
Ferric oxide (Fe_2O_3)	0.80
Lime (CaO)	0.05
Magnesia (MgO)	0.30
Alkalies	0.31
	100.12

In Colorado and Montana both quartzite and sandstone are used to some extent in the manufacture of silica brick.

METHOD OF MANUFACTURE

Quarrying

The methods of quarrying as necessitated by the mode of occurrence of the quartzite are so obvious as to need no description. A point that will bear emphasis however, is the necessity of adequate supervision in the selection of the rock, whether it be from floes or from the solid measures. Whatever the location or occurrence of the rock, such care is essential; the easiest way to keep bad rock out of the brick is never to let it reach the plant.

Grinding

After the rock broken to one man size is received at the rear of the brick plant, it is put through some one of the familiar types of crushers, which reduces it to practically the size of a 1- to 2-in. ring. It is then elevated to storage bins from which it may be fed direct in definitely weighed quantities to the wet pans for grinding.

Into the wet pans, as the grinding proceeds, 2 per cent. of lime, by weight, is introduced as milk of lime. When the brick is burned, the lime in combination with the other elements in the mass serves to give the necessary bond or physical strength. The quality of the lime has no small effect upon the resulting characteristics of the brick. The effect of an increase in lime from 2 to 3 per cent. is not marked; but such as it is, renders the brick less able to withstand abrasion, as is indicated by rattler tests. The decrease in strength on further additions of lime up to 7 per cent. is progressive, and becomes more and more evident with each addition. There is, on the other hand, a decrease in strength at 1.5 per cent. of lime, and a marked weakness and friability as the amount of lime is still further diminished. As regards ultimate refractoriness, no marked effect is noticeable upon increasing the lime addition up to 3 per cent. From that point on, the effect is clear in a lowering of the fusion point.

The fineness to which the charge in the wet pan is to be ground depends upon the nature of the brick to be made from it and is, of course, controlled by the time during which the charge is allowed to remain in the pan. In general, one charge, containing material for 200 brick, will be ground at the rate of approximately four pans per hour for standard 9-in. brick, while to obtain the fineness requisite for the intricate and difficult shape-work necessary in byproduct coke-oven construction, but two pans per hour may be possible. The rock is exceedingly hard, and a comparatively slight increase in the fineness of grind is accompanied by a seemingly disproportionate lowering of grinding capacity, and increase in wear on the pans.

The actual differences in grind, however, are much smaller than is often imagined. Figs. 1 and 2 are photomicrographs with a magnification of 2 diameters, the ground material showing on a background of cross-section paper with 0.1-in. spacing. The specimens are from pieces of green or unburned silica brick of equal weight, crushed down after drying. Fig. 2 indicates typical fine-ground shape-work, while Fig. 1 is that of average standard 9-in. brick.

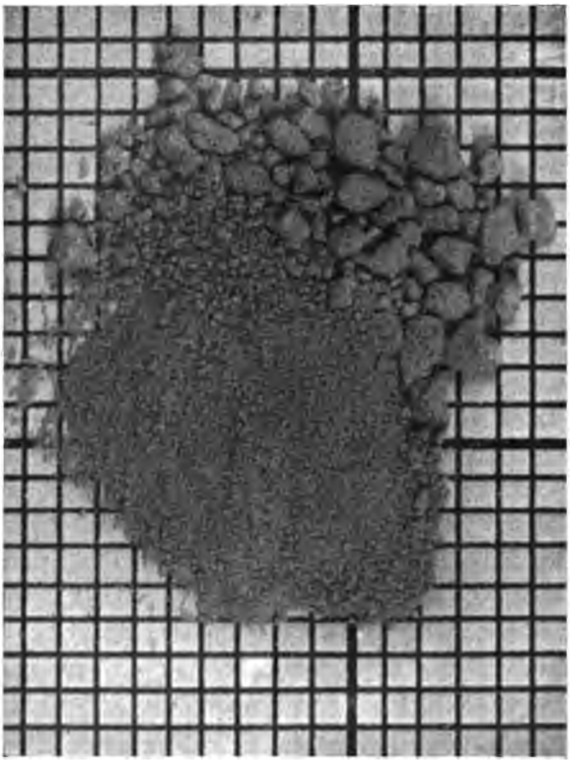

FIG. 1.—MATERIAL FROM UNBURNED AVERAGE STANDARD 9-IN. BRICK. MAGNIFICATION 2 DIAMETERS.

It will be plainly apparent from the photomicrographs that the difference lies not so much in the actual size of either the coarse or the fine grains as in the relative proportion of fines. In the finer-ground brick the larger proportion of these is clearly evident. There can be no question that the general effect of increasing the fineness of grind is to improve the abrasive resistance of the brick, if it is properly burned. In general the requirements as to grind contained in typical coke-oven specifications have worked no hardship upon the manufacturer; for the present-day standard of workmanship on byproduct oven shapes is such as to "carry its own grind with it," if the expression may be used. To secure the accurate filling out of the shapes, and the clean-cut accuracy everywhere essential, the grind is actually much finer than is requisite to satisfy the usual specifications as to that particular feature.

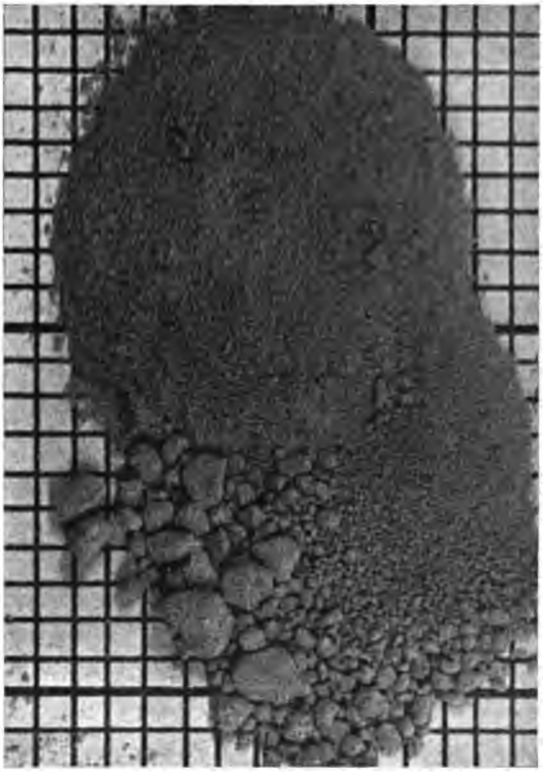

FIG. 2.—SAMPLE SHOWING MATERIAL USED IN TYPICAL FINE GROUND SHAPE WORK MAGNIFICATION 2 DIAMETERS.

Molding

The ground silica material as ready to mold is of about the consistency of damp sand, but probably because of the extreme angularity of its grains has less flow as it is manipulated. As a consequence, the molds must be filled by pounding with heavy plank beaters covered with metal, and all corners and shapes of irregular outline must be rammed in by hand, much as sand molds are made, but with much heavier blows. Unequal ramming, which leaves too great a variation in the density of the green brick, will give unequal expansion in the kiln. It is by no means uncommon to lose hundreds of a new and difficult shape through such fire-cracking. To get proper molding and ramming requires the strictest supervision.

The wear on the molds due both to the attrition of the grains and the pounding is exceptionally severe. In view of the extreme accuracy required, steel molds are used exclusively for all high-grade coke-oven work. These molds are often decidedly complicated, and always expensive, as "loose sides" must be used throughout. Ground silica has, as molded, almost no physical strength, and can be taken from the mold only by allowing the sides to come with it, after which the sides are stripped.

After molding, the shapes are placed on pallets, on which the brick are dried either on the steam-heated hot floor or on rack cars in some form of tunnel dryer (Fig. 3). The brick are then ready to be set in the kilns. Although setting is a most important feature in the manufacture, no attempt will be made to discuss it here.

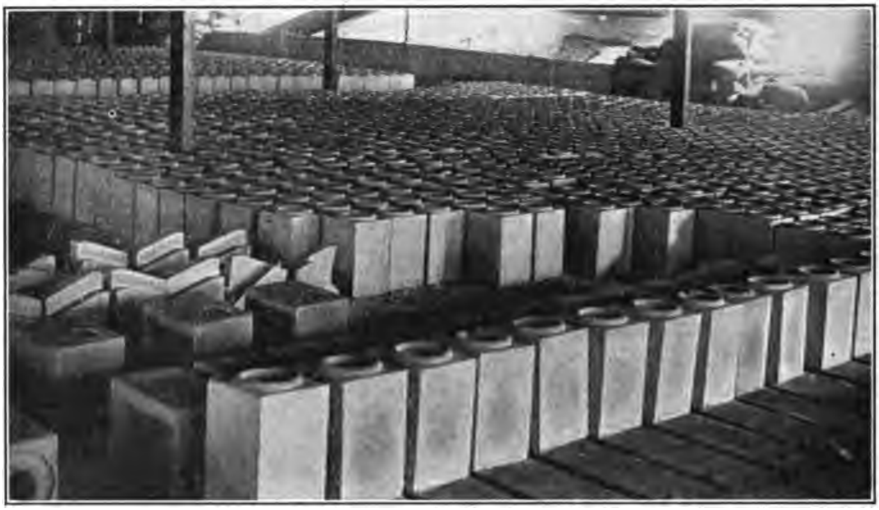

Fig. 3.—Typical Section of Hot Floor, showing Coke Oven "Gun" Brick-shapes in Process of Drying.

Burning

The vast majority of silica brick are burned with soft coal in the familiar type of round, down-draft kiln, varying in diameter from 30 ft. upward. Although practice may vary somewhat, and much depends on the size of the kiln, the following schedule of burning may be considered as fairly normal. Assume that the setting is complete and the doors are sealed. Fire is immediately started, and continued slowly and with slight acceleration for from 9 to 12 days, during the last 2 or 3 of which the final temperature should be practically attained. The kiln is then held at the required temperature, say at 26 cone (1,650° C., or

3,002° F.) directly in front of the firebox, for another day, after which the fireboxes may be sealed and allowed to remain thus for approximately 24 hours more. Cooling will require about as long as the burning proper; but this varies somewhat with atmospheric conditions.

It is obviously difficult to take temperatures in the interior portions of the kiln except by cones; but there are often most perplexing variations in this method. There are apparently certain conditions in the atmosphere of a burning silica kiln which alter the readings of standard cones to no small degree. Cones which give concordant results, week in and week out, in clay kilns are often decidedly erratic in registering the temperatures of silica kilns. It is by no means uncommon to witness cones of higher number go down, while next to them those lower in the series are either but slightly bent or apparently unaffected.

Effect of Burning on Composition and Expansion

In any discussion of the desirable qualities of silica brick there is frequently expressed the necessity that "all the expansion shall be burned out of them." This brings out the following questions: What is the nature of this expansion; what produces it; what is its extent, and how fully can it be controlled.

That the terms used in the discussion may be clear, let it be premised that all silica brick are of course made in molds smaller than the proposed size of the finished or burned brick, and that under the influence of heat a permanent expansion occurs, swelling the brick from the mold or "green" brick size to the required or "burned" size. It is desired that this permanent expansion occurring during burning be completed so as to preclude further permanent expansion in subsequent use.

Now, in every burned silica brick expansion will occur on heating—a true thermal expansion, identical in its general character with that incident to the heating of practically all bodies. Such thermal or temporary expansion will disappear upon cooling, and being the expression of the coefficient of expansion of the silica it is unavoidable. This thermal or temporary expansion; occuring upon heating a burned silica brick is not to be confused with the permanent expansion produced in the body of the brick during its manufacture. It is argued that, if a silica brick is not properly burned, then, upon reheating in actual coke-oven practice, there will be not only a normal thermal expansion but an additional expansion of indeterminate amount which will be permanent. It is essential in the construction of the coke ovens that the expansion, whatever its nature, be as small as possible, and that, whatever it is to be, it be known and adequately cared for.

It will be recalled that the silica brick under consideration is formed by adding 2 per cent. of lime to quartzite rock of the following composition, say:

	Per Cent.
Silica (SiO_2)	97.80
Alumina (Al_2O_3)	0.90
Ferric oxide (Fe_2O_3)	0.85
Lime (CaO)	0.10
Magnesia (MgO)	0.15
Alkalies	0.40
	100.20

The representative silica brick produced may be assumed to have the following composition:

	Per Cent.
Silica (SiO_2)	96.25
Alumina (Al_2O_3)	0.88
Ferric oxide (Fe_2O_3)	0.79
Lime (CaO)	1.80
Magnesia (MgO)	0.14
Alkalies	0.39
	100.25

What are the changes which occur as the siliceous mass is burned?

Certain of these are obvious, such for instance as the formation of a bond of one or more of the lime-silica compounds such as the meta-silicate CaO, SiO_2 or the ortho-silicate $2CaO, SiO_2$. With the exception of the lime present, the ferric oxide of iron and the alumina, though small in quantity, are the preponderating impurities and there is undoubtedly formed a certain amount of silicate of iron as well as compounds of the lime-alumina series. Whereas the lowest melting point incident to any eutectic of the lime-silica system is 1,436° C. (2,617° F.) and of the alumina-silica series is 1,610° C. (2,930° F.), while that of the lime-alumina system is 1,400° C. (2,552° F.), some of the eutectics of the ternary system CaO, Al_2O_3, SiO_2 melt at approximately 1,170° C. (2,138° F.) and are thus among the first of the flux compounds resulting.

No less important, though less recognized, changes are occurring in the brick as it is heated. The evidence of the change, regardless of its reason, is easily indicated by the most casual measurement when brick are drawn from the kiln from time to time, as higher temperatures are reached. The final expansion will be very close to 3/8 in. per foot.

The theory of this expansion is that a change occurs in the crystalline form of the silica, an inversion from the quartz crystal to the crystalline form of crystobalite. The intermediate crystalline form of tridymite does not apparently occur here. Since these various crystal forms have varying specific gravities the volumes accordingly vary.

The valuable work done by Day and Shepherd[1] and particularly by Fenner,[2] give the results in a large field of laboratory experiment, which make possible a fair understanding of what occurs in the actual burning of the silica brick. Fenner's work was to establish among other things the relations between the silica minerals quartz, tridymite, and cristobalite, any of which may be converted into any other of these by appropriate heat treatment. For the purposes of this discus-discussion it is unnecessary to touch upon the alpha and beta phases of these minerals. In substance, the results under the conditions imposed by the experiments were that powdered quartz, if heated, remains quartz until a temperature of 870° C. (1,598° F.) is reached, at which inversion point there begins a slow change to tridymite. If heat is still applied, there is no further change from tridymite until 1,470° C. (2,678° F.) is reached, when inversion to the form of cristobalite has its beginning. There is no further change until the cristobalite melts into a glass. Thus the stable phase of quartz (quartzite rock as applied to silica-brick manufacture) extends to 870° C. (1,598° F.). Between 870° C. (1,598° F.) and 1,470° C. (2,678° F.) the stable phase is tridymite, while above 1,470° C. (2,678° F.) cristobalite is stable.

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[1] The Lime-Silica Series of Minerals, *American Journal of Science,* 4th series, vol. xx, No. 130, pp. 265 to 302 (1906).

[2] The Stability Relations of the Silica Minerals. *American Journal of Science,* 4th series, vol. xxxvi, No. 314 (1913).

As a whole it must be carefully noted that the changes from one mineral to another were exceedingly slow, particularly near the inversion points. So slow in fact were the alterations, that the melts were made with a large excess of sodium tungstate which acted as a flux or catalyzer. The powdered quartz or whatever might be the silica mineral under study was stirred into the molten bath of flux and the platinum container was then heated in a suitable furnace. This flux in no way entered into chemical combination with the silica but only aided and quickened its action, apparently furnishing a fluid medium through which the molecular rearrangement was facilitated. In spite of this, the slowness of the inversion, often requiring many hours, will bear emphasis in drawing conclusions regarding any parallelism between these experiments and the burning of brick.

A particularly noteworthy fact is that in the absence of a flux or catalyzer, the inversion from quartz to tridymite does not occur, but the inversion is direct to cristobalite, and there is a shift in the inversion point to as low as 1,250° C. (2,282° F.) although the change at this temperature is slow.

This entire absence of tridymite in quartz heated well above 870° C. (1,598° F.) the quartz-tridymite inversion point, although difficult of explanation, is apparently proven. As indicative of the time element requisite for such an inversion, in one experiment fine-ground quartz heated 108 hr. at 1,250° C. (2,282° F.) without flux gave but a small percentage of inversion while with 90 hr. at 1,360° C. (2,570° F.) approximately two-thirds became cristobalite.

In considering these changes in the crystalline form of silica, some of the characteristic properties of the three will not only make clearer the method of distinguishing them, but also throw some light on the reasons for the effects these inversions exhibit.

Quartz has a specific gravity of 2.65, tridymite of 2.27, and cristobalite of 2.33. Each has its characteristic crystal form, the quartz being usually in the familiar hexagonal pyramids, prisms or combinations of these. Tridymite often occurs in thin tablets or plates hexagonal in shape. Cristobalite is exceedingly irregular in the development of its crystal form but often develops octahedral caps along the closely interlacing axes of elongation.

It is the difference between the indices of refraction of these minerals, however, that furnishes the greatest aid in distinguishing the one from the other.

Tests.—As a starting point from which to work in determining the effect of burning, a set of sample silica brick were taken from one kiln, all from the same zone and from a kiln of absolutely normal burn. In this instance they were first burned at a point about 8 ft. from the wall and 6 ft. from the bottom. This sample was divided into three parts, one to be kept unaltered, one to be reburned under the same conditions, and a third to have two reburns. In addition to this, pieces of quartz rock received one and two burns respectively, with the view of obtaining the same data.

The primary object was to determine, if possible, how far the inversion from quartz to cristobalite had progressed under the successive conditions. Each burn was made as nearly uniform as practicable, and in each succeeding burn the effort was made to maintain identical conditions. At that part of the kiln where these brick were burned the temperature reached 1,540° C. (2,804° F.), and they were subjected to that heat for approximately 40 hr.

* In general, the chief optical constant serving to distinguish cristobalite on the one hand from quartzite and calcium silicate on the other was the index of refraction. The slow ray of quartz has a refractive index of 1.544 while the index of the fast ray of cristobalite is about 1.485. The immersion method was used to determine the indices of the grains, the powdered brick being immersed in a liquid of index 1.51. By this method it was possible to distinguish between the higher-refracting quartz and calcium silicate and the lower-refracting cristobalite.

With the aid of a *camera lucida,* the fields were sketched on very thin tracing paper. The grains, as represented on the paper, were then cut out and weighed for each constituent. Aside from the fact that the composition of different fields may vary, other errors are entailed in the sketching, cutting out and weighing so that a variation of 5 per cent. is justifiable. In these determinations the fields of the same specimen checked to within 5 per cent. maximum and 1 per cent. minimum error.

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* The heartiest appreciation is here expressed to A. A. Klein, Petrographer in the U. S. Bureau of Standards at Pittsburgh, for his courtesy in making these determinations that they might be concluded in time for presentation here. The quality of Mr. Klein's work needs no comment.

Results.—The tabulated data are as follows:

Specimen	Field	Quartz and Calcium Silicate, Per Cent.	Average, Per Cent.	Cristobalite, Per Cent.	Average, Per Cent.
Lot 1	1	24.87	....	75.13	....
(1st burn)	2	20.42	22.65	79.58	77.35
Lot 2	1	14.31	....	85.69	....
(2nd burn)	2	19.96	17.13	80.04	82.87
Lot 3	1	17.44	....	82.56	....
(3rd burn)	2	14.60	16.02	85.40	83.98
Quartz rock	1	50.08	....	49.92	....
(1st burn)	2	51.05	50.57	48.95	48.95
Quartz rock	1	32.70	....	67.30	....
(2nd burn)	2	30.06	31.38	69.94	68.62

A number of interesting deductions may be made even from these preliminary tests. The maximum change takes place at the first burning, followed by a further appreciable inversion at the second, and practically no change at the third.

The grain structure of the reburned silica brick may throw some light on this subject. Here the grains may be seen to be essentially cristobalite, with the quartz occurring as an inclusion. The last stages of the inversion are apparently exceedingly slow within the protecting envelope.

The smaller percentage of inversion in the case of the quartz rock as compared with the brick is very noticeable, and is undoubtedly due for the most part to the grinding of the quartz in the manufacture of the brick. This is wholly in line with Fenner's laboratory experiments, in which he found the influence of comparative grinds to be slight if the silica mineral were heated with a flux, but to be marked if the material were heated without a flux. Fine grinding greatly hastened the inversion. The greater freedom of movement in expansion as compared with the exceedingly dense quartz body is probably the controlling factor, rather than any difference in the penetration of the heat.

The part played in the inversion (if any part is played) by the lime and its compounds present, is of much interest. It appears logical that the greater fluidity conferred by the fluxes present should have an influence; but it must be borne in mind that, as compared with previous experimental work on silica minerals, the amount of flux present in the brick is exceedingly small, and that definite chemical compounds are formed in combination with the brick body. In Fenner's work the sodic tungstate formed no such chemical compounds, and remained unchanged at the conclusion.

The fact moreover that no tridymite was determinable in either the burned brick or the calcined quartz is explained by the presence of the negligible amount of flux. This would favor the direct inversion of quartz to cristobalite.

Thus far, we have assumed the changes in crystal form to be desirable, because of their effect on the expansion of the material.

Effect of Burning on Physical Strength

It is also of interest to note the effect of the reburning on the physical strength of the brick. To determine this, the following tests were run:

Cross-Breaking or Modulus of Rupture Test

Number of Burns	Mark	Breadth of Brick, Inches	Depth of Brick, Inches	Weight Causing Fracture of Brick, Pounds	Length between Supports, Inches	Modulus of Rupture, Pounds Per Square Inch	Average, Pounds
1	9 A	2.46	4.40	3,550	6	671	....
1	9 A	2.42	4.40	3,000	6	576	624
2	9 B	2.50	4.50	4,140	6	730	....
2	9 B	2.50	4.50	5,000	6	889	809
3	9 C	2.06	4.50	4,500	6	971	....
3	9 C	1.96	4.50	4,550	6	1,031	1,001

Crushing-Test: Whole Brick Crushed Flat

Number of Burns	Mark	Dimensions, Inches	Area, Square Inches	Load, Pounds	Crushing Strength per Square Inch, Pounds	Average, Pounds
1	9 A 2	9.00 by 4.40 by 1.96	39.60	176,800	4,465	....
1	9 A 2	9.04 by 4.44 by 2.04	40.14	167,000	4,161	4,313
2	9 B 2	9.10 by 4.46 by 2.02	40.58	186,500	4,596	....
2	9 B 2	9.10 by 4.40 by 2.06	40.04	168,000	4,196	4,396
3	9 C 2	9.06 by 4.42 by 2.00	40.04	195,000	4,870	....
3	9 C 2	9.12 by 4.46 by 2.06	40.68	168,000	4,130	4,500

The increased strength of the brick is evident.

CONCLUSIONS

In considering whether or not all the expansion possible is burned out of the brick, it is necessary to determine first the limits and then how closely they are approached.

Day, Sosman and Hostetter say[3] that the true specific volume of cristobalite as compared with quartz at 20° C. is 13.4 per cent. greater.

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[3] The Determination of Mineral and Rock Densities at High Temperatures. *American Journal of Science,* 4th series, vol. xxxvii, No. 217, pp. 1 to 39 (1914).

Assuming a green brick to be wholly pure quartz (100 per cent. SiO_2), and the brick burned from it wholly cristobalite, the volume increase would then be 13.4 per cent. if the relative porosity of the body be disregarded. This theoretically possible expansion is diminished by the effect of the impurities present in the actual brick, and by the resulting compounds formed. For instance, 4 per cent. of impurities would produce a much larger proportion of compounds to which this expansion phenomenon does not apply.

An expansion of, say, 3/8 to 7/16 in. per foot, to which byproduct coke-oven silica is burned, thus provides for somewhat more than 13 per cent. increase in volume.

It must be borne in mind, however, that by quick burning the brick may be easily expanded by double or treble the normal amount; but sound brick do not result. This excessive expansion is due to minute fissures opened up in the body of the brick by a too sudden rise in temperature during burning. In the early stages of firing the heat must be raised very slowly. In general the conclusion is justified that the present good commercial practice carries the results of the quartz-cristobalite inversion to the greatest degree economically possible and that the measure of the expansion of the brick affords a closely approximate measure of the inversion—providing only that the brick are sound.

The short series of tests outlined above are given with full realization that, while indicative, they are by no means conclusive. They were but the preliminary runs, necessarily made somewhat in haste, but giving data which permit intelligent plans to be made for the more comprehensive tests now under way. There is much to be learned as to the effect of both time and temperature as well as the influence of pressure, which may produce a shifting of some of the inversion points. Judging from the work thus far done, it is impossible to expect the full inversion of quartz to cristobalite under any practical operating conditions of manufacture.

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Mine Pumping

BY CHARLES LEGRAND,* DOUGLAS, ARIZ.

(San Francisco Meeting, September, 1915)

THE problem of mine pumping is so much affected by local conditions, and those conditions are so liable to changes during the life of a mine, that the best system to use is difficult to determine. The experience of the writer has been that, in general, for copper mines, electric pumping is most satisfactory unless the quantity of water to be pumped is great or the mine does not use electric power for any other purpose. As a rule, however, figuring on the cost of necessary power plant, the total cost of installation is greater with electric pumps than steam pumps.

As the water is liable to be gritty, outside-packed plunger pumps are to be recommended, and if the water is not acid, or only slightly so, chilled cast-iron plungers pay for their extra cost very quickly in diminished cost of packing. With chilled plungers it is possible to use metallic packing instead of soft packing, if the water is not too gritty nor the lift too high.

In vertical pumps this packing is satisfactory for lifts up to 400 ft., in the experience of the writer, and may be found satisfactory for higher lifts.

For water slightly acid, cement-lined pump bodies have given good satisfaction.

For acid water, both plungers and pump bodies should be made of acid-resisting bronze.

For high lifts, especially with gritty water, a satisfactory pump valve is difficult to find. To reduce the unbalanced pressure on the valve at time of opening it is advisable to have narrow seats; this, however, brings high pressures on the seats when the valve is closed. On large valves for lifts of 600 to 1,000 ft. we have used leather, hard rubber, vulcanized fiber, and soft brass; for clear water the latter has given best satisfaction.

The difficulties due to grit make it advisable to have large sumps where water can settle, and if possible, to have two of them so that they can be cleaned alternately.

At the mines of the Old Dominion Copper Mining & Smelting Co., Globe, it was found that a small air ejector discharging into a mine car is a very convenient way to clean mud and sand from a sump.

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* Consulting Engineer, Phelps, Dodge & Co.

The steam pumps can be divided into two broad classes: The direct acting, with simple, compound, or triple-expansion steam cylinders; and fly-wheel pumps with compound or triple-expansion steam cylinders. All the steam pumps are usually run condensing, both for steam economy and because exhaust steam cannot be discharged in the mine.

The steam consumption per water horsepower diminishes in the order given above and the complexity of the pumps increases as the steam consumption diminishes.

The simple direct-acting pump is so uneconomical in steam consumption that it should not be used for permanent pumping although very simple and convenient in emergencies. It can also be used with compressed air instead of steam.

The compound direct-acting pump for small power is a satisfactory pump.

The triple-expansion direct-acting pump is fairly economical in the use of steam if proportions of steam and water ends are correct for the lift. In a great many installations this is not the case and the steam end of the pump is too large for the work done, increasing considerably the steam consumption per horsepower. It is quite usual for mine managers to specify a higher lift for the pump than they expect to have when the pump is installed, so as to enable them to put the pump at a lower level later on; if a pump has to be ordered for such a change of lift it will be found advantageous to order the pump for the correct lift but specify the water end strong enough to stand the maximum pressure expected in the future. When lift on pump is increased the size of plunger can be altered at small cost to keep the correct proportions between steam and water end. This naturally decreases the capacity of the pump in proportion to the lift

All the direct-acting pumps take comparatively little room and are easy to move from level to level.

The compound and triple-expansion fly-wheel pumps are more economical in use of steam, but are much more expensive of installation, require more room, and must be installed on good foundations that will not move or move as a whole. Their use can generally only be justified if the quantity of power to be delivered is considerable, steam expensive, and the probable life of the mine will repay the difference in cost.

Whether a surface condenser or a jet condenser should be used depends an local conditions. The surface condenser has less moving machinery and uses less steam than the jet condenser, but is liable to be out of service a longer time while tubes are being cleaned.

The question of proper size of steam line between boilers and pumps is one which usually is not given sufficient attention, most steam lines being too large for maximum economy. The writer has found in most cases that it is advantageous to have a steam pipe which gives a drop of pressure of 5 to 10 lb. between boilers and pumps, as the reduced condensation pays for the loss of pressure. When a small pipe line is used the usual separator at the pump should be made a separator-receiver, to steady the flow through the pipe line.

All steam pumps have a great advantage where flow is variable, as the speed and delivery of pump are easily adjustable to varying conditions.

The electric pumps can be divided in two broad classes: The plunger and centrifugal pumps.

The plunger pumps are of many designs; the writer generally prefers the vertical type as giving more even wear on packing and plungers and taking less floor space, although they require more head room.

For high lift the quintuplex pump having a practically steady flow on the discharge is advantageous, as it can be used without air chamber.

The motors are generally geared to the pump through a single or double reduction of gears. If straight spur gears are used it is not advisable to have a single reduction of more than 8 to 1, and slow-speed motors have to be used. If herringbone gears are used single reduction of larger ratio is permissible, allowing a motor of standard speed for pump drive. The herringbone gear is more efficient than the spur gear, but requires much closer adjustment of pitch line, so that bearings have to be made adjustable to maintain proper distance between pinion and gear shafts. For this style of drive the proper design of bearings is very important.

A flexible coupling between pinion and motor shaft usually pays for itself by reduced maintenance of motor, although on pumps using 50 hp. or less, a motor with outboard bearing, and either a fabric or raw-hide pinion geared directly to pump without coupling, has been found quite satisfactory.

With alternating-current motors, unless special motors are used, it is necessary to unload the pump at start by means of a bypass valve. Motors with high starting torque can be obtained, but are less efficient than standard motors, and are not necessary unless the pump is to be used with an automatic starter.

Direct-current motors will start the pump under full head, with a starting rheostat made for this service.

The writer has had little experience with centrifugal pumps. As a rule, however, they have the advantages of great simplicity, and ease of connection to motor, having no gearing; they are, however, less efficient than plunger pumps and have to be designed for exactly known conditions. They are sensitive to change of motor speed or changes of head, in a high-lift pump of moderate capacity the passages are small, the water travels at high speed, and if at all gritty the wear upsets the proportions of the pump and quickly affects the efficiency.

With plunger pumps the amount of water delivered can only be altered by varying the speed of the motor, starting and stopping the pump as required by the flow of water, or by passing a portion of the water to the pump suction.

The first method is easy where direct current is available, but more complicated or less efficient where alternating-current motors are used.

The second method can be used with both systems of power, but is hard on the power plant if pumping load is a large proportion of the total.

The third one is inefficient in the use of power, as the pump is working at full load at all times.

As the efficiency of electric pumps is not much affected by their size, where flow of water is variable it is advisable to have more units of a smaller capacity so that the number of units running can be adjusted approximately to the quantity of water pumped and one unit started and stopped by hand or automatically to take care of the variation of water level in the sump.

Electric pumps usually require less attendance than steam pumps and reduce the quantity of heat liberated in the mine, which is a considerable advantage in hot mines.

The electrical measurements are so much easier to make than the steam measurements that an electric pump is likely to be kept at a higher efficiency. With an electric pump it is easy to show the attendants the evil effects of packing the glands too tight and the large amount of power that can be thus consumed without heating the plungers.

The electric plunger pumps can be operated at a lower lift than they are built for with very little loss of efficiency.

Where the proper submergence can be obtained an air lift is inexpensive to install and will handle very great quantities of water in a small space, and although the cost of the air is comparatively great, there are practically no other running costs. This method cannot be used from the lowest level of a mine, without a lot of complications, as there is then no way to get the proper submergence.

Air lifts were used at the mine of the Old Dominion Copper Mining & Smelting Co. at Globe in a recent emergency when water flow increased on the upper levels of the mine and got beyond the pumping capacity of the plant. A 10-in. air lift raising the water 200 ft. (exclusive of friction), using air at 90 to 95 lb. pressure at the power plant, required the following amount of air (measured by flow meter) at a barometric pressure of about 27 in. of mercury.

Gallons Water per Minute	Cubic Feet Free Air per Minute	Cubic Feet Free Air per 1,000 Gal.
1,011	1,353	1,338
1,680	1,809	1,080
1,794	2,262	1,261
1,925	2,658	1,375
1,965	3,219	1,638

Another lift raising the water 431 ft. exclusive of friction gave the following results:

Gallons Water per Minute	Cubic Feet Free Air per Minute		Cubic Feet Free Air per 1,000 Gal.	
1,122	3,051		2,718	
1,233	3,306	3,395	2,681	2,753
1,233	3,484		2,825	
1,291	3,832	3,875	2,968	3,002
1,291	3,919		3,035	
1,325	4,089		3,086	

In these tests, which were kept up from 1 to 2 hr. each, the measurements of air were taken with a General Electric Co. air-flow recording meter installed carefully and checked after test according to instructions furnished by the makers; the results are probably correct within 5 per cent.

The measurements on water were obtained by taking the reduction of speed of the pumps when air lift was working and the water level in sump kept constant and a volumetric efficiency of 90 per cent. assumed for the pumps. All pumps were equipped with revolution counters.

The average speed of the pumps was recorded before and after the air test to insure that the water flow had not changed in the mine. The quantities of water are very nearly correct.

The above figures are not given as absolute, but only to give a rough idea of what can be done with an air lift. In both cases the submergence was about 190 ft.

In the following table are given the results of some tests on various sizes of steam and electric pumps. These tests were taken under running conditions and for steam pumps the water fed to boilers was taken as used by the pumps, all the auxiliaries of the boiler plant being run from another source of supply.

Tests 1, 2, and 3 were taken by condensing the exhaust of the pump and weighing the condensate and estimating the volumetric efficiency of water end at 90 per cent., as the delivered water was not measured.

Tests 4 to 9 were not taken by the writer.

Tests 10 and 11 are on triple-expansion four-cylinder pumps with two low-pressure cylinders. On these tests steam was superheated 35° when leaving boilers and the pumps were tested together.

Tests 12, 13, and 14 were on pumps immediately after starting.

Test 15 on same pump as No. 14 after two weeks' run.

Test 16 is on single-reduction spur-gear pump.

Test 17 is on single-reduction herringbone-gear pump.

Test 18 is the best test on air lift for 200 ft.

Test 19 is the next to the best test on same.

Test 20 is the best test on air lift for 431 ft.

Test 21 is the next to the best test on air lift for 431 ft.

On Tests 18 to 21 the steam consumption per water horsepower-hour is based on air compressors requiring 38 lb. of steam per 1,000 cu. ft. of free air, delivered compressed to 9 lb. gauge pressure.

Steam Pumps

Test Number	Kind of Pump	Steam Cylinders, Inches	Water Plunger, Inches	Stroke, Inches	Speed, Rev. per Min.	Water Del'd During Test, Gal. per Min.	Lift Including Friction, Feet	Steam per Water Horsepower-hour	Boiler Pressure, Pounds
1	Single Duplex............	9	8¼	10	33.0	270	120	98.0	135
2	Single Duplex............	14	8½	10	33.0	285	111	135.0	135
3	No. 1 Exhausting in No. 2 as a Compound pump....	....	....	....	....	350	120	78.0	135
4	Triple Expansion Duplex...	10–15½–26	8	24	14.1	278	591	61.5	122
5	Triple Expansion Duplex..	15–23–39	10	24	12.7	391	926	49.4	125
6	Triple Expansion Duplex..	15–23–39	10	24	21.7	658	629	42.1	131
7	Triple Expansion Duplex..	18–28–47	12	24	13.0	584	1,056	35.9	143
8	Triple Expansion Duplex..	18–28–47	12	24	25.7	1,207	1,056	27.8	142
9	Cross-Compound Fly Wheel	19–35	5–⅝	36	32.4	502	1,059	23.7	110
10	Triple Expansion Fly Wheel	14–26–26–26	7–5/16	36	38.0	878	652	18.0	137
11	Triple Expansion Fly Wheel	14–26–26–26	6–⅜	36	32.0	553	883	18.0	137

Electric Pumps

	Kind of Pump	Motor, Horsepowe	Water Plunger, Inches	Stroke, Inches	Speed, Rev. per Min.	Water Del'd During Test, Gal. per Min.	Lift Including Friction, Feet	Steam Based on 25 lb. per Kw-Hr.
12	Vertical Triplex Double Reduction	10	4	9	35	51	475	32.7
13	Vertical Triplex Double Reduction	20	4	9	37	54	1,162	28.1
14	Vertical Triplex Double Reduction	50	9	10	37	302	423	29.1
15	Vertical Triplex Double Reduction	50	9	10	37	303	423	25.6
16	Vertical Quintuplex Single Reduction	110	4½	12	40	159	1,155	23.1
17	Horizontal Quintuplex Single Reduction	250	9	18	52	1,245	605	22.2
				Air Lifts			Excluding Friction	
18	....	....	....	....	....	1,680	200	48.7
19	....	....	....	....	....	1,794	200	57.5
20	....	....	....	....	....	1,122	431	57.0
21	....	....	....	....	....	1,233	431	57.6

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Conveyor-Belt Calculating Chart

BY J. D. MOONEY AND D. L. DARNELL,* AKRON, OHIO

(San Francisco Meeting, September, 1915)

THE accompanying chart has been drawn for the convenience of engineers as a means of quickly determining the correct number of plies of conveyor belts operating under specific conditions.

The calculations are based on the average safe strength (factor of safety, 15) of the various standard rubber conveyor belts.

The calculations assume maximum loading conditions; that is, the belt is considered as carrying the greatest load that it will handle without spillage at ordinary belt speeds. This not only produces the most economical operating conditions, but also the maximum tension in the belt.

The chart is a graphical representation of the formula:

$$p = kgW(L + 10H)$$

Where, p = the correct number of plies
k = a constant, depending on the type of drive
g = the weight in pounds per cubic foot of material handled
W = the width of the belt in inches
L = the length of the belt in feet (approximately twice the center distance).
H = the difference in elevation between the head and tail pulleys, in feet.

For a simple drive, with a bare pulley, $k = \frac{1}{250,000}$
For a simple drive, with a rubber-lagged pulley, $k = \frac{1}{300,0\ 0}$
For a tandem drive, with bare pulleys, $k = \frac{1}{375,000}$
For a tandem drive, with rubber-lagged pulleys, $k = \frac{1}{455,000}$

The chart is drawn for a simple drive, with a bare pulley $\left(k = \frac{1}{250,000}\right)$; therefore, the number of plies obtained from the chart should be multiplied by the factor 0.83 or $\frac{5}{6}$ for simple, lagged drive; the factor 0.67 or $\frac{2}{3}$ for tandem, bare; and the factor 0.55 or $^{11}/_{20}$, for tandem, lagged.

The formula $p = kgW(L + 10H)$ has been developed mathematically

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* B. F. Goodrich Co.

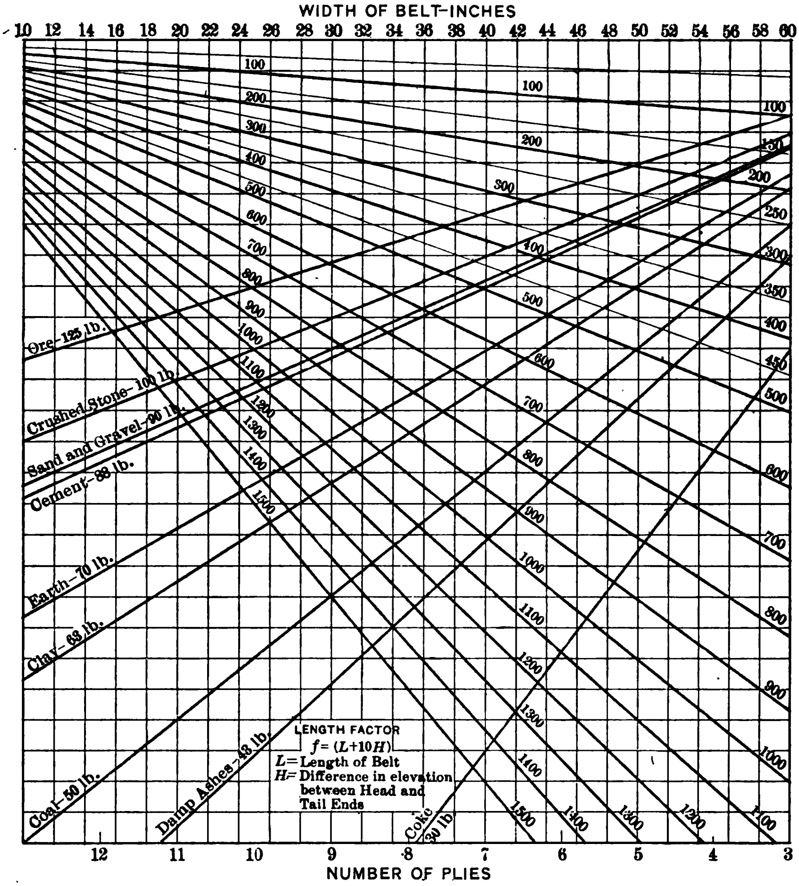

FIG. 1.—CONVEYOR BELT CALCULATING CHART.

from the following formulas, which have been found to work very satisfactorily in average good practice:

$$U = \frac{0.08W^2Sg}{5000}$$

$$HP = \left(\frac{0.02l}{100} + \frac{0.01H}{10}\right)U$$

$$T = \frac{C \times HP \times 100}{S}$$

$$p = \frac{T}{24W}$$

Where, U = Capacity in tons per hour
W = Width of belt in inches
S = Belt speed in feet per minute
g = Weight per cubic foot of material handled
HP = Horsepower developed in driving conveyor belt
l = Length of the conveyor, in feet (approximately $\frac{1}{2}L$)
H = The difference in elevation between the head and tail pulleys, in feet
T = The total tension in the belt, in pounds
p = Correct number of plies
C = The constant of the drive.

For a simple drive, bare pulley, $C = 600$
For a simple drive, rubber-lagged pulley, $C = 500$
For a tandem drive, bare pulleys, $C = 400$
For a tandem drive, rubber-lagged pulleys, $C = 330$

The length factor, $f = (L + 10H)$, represented on the chart by the lines 500, 600, 700, etc., is a developed factor equal to the sum of the length of the belt and ten times the difference in elevation between the head and tail pulleys.

To find the correct number of plies for a conveyor belt, knowing the width, the length, the difference in elevation between the head and tail ends, and the kind of material to be handled:

Start from the width given at the top of the chart and move down until this line intersects the line corresponding to the proper length factor; then move either right or left until the line corresponding to the given material is met; then move down again to the scale of plies, where the next largest figure will give the correct number of plies.

For example: To find the correct number of plies for a conveyor belt 36 in. wide and 300 ft. long, with 20 ft. difference in elevation; handling sand and gravel:

Follow the line from the 36-in. width downward until it intersects the 500 length factor line; then follow to the right until the sand and gravel line is intersected; then down to the ply scale, where the ply will be found to be 7.

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Ventilation of the Copper Queen Mine

BY CHARLES A. MITKE,* BISBEE, ARIZ.

(San Francisco Meeting, September, 1915)

INTRODUCTION

THE Copper Queen mine is composed of seven divisions which are operated through the following shafts:

Division No.	Shaft	Depth, Feet	Air Current
1	Uncle Sam	600	Downcast (shut down April, 1914, to curtail production)
	Southwest	600	Downcast
2	Czar	400	Downcast
3	Holbrook	600	Downcast
4	Spray	800	Downcast (shut down April, 1914, to curtail production)
5	Gardner	1,000	Downcast
6	Dallas	1,400	Downcast
	Lowell	1,600	Upcast
7	Sacramento	1,700	Downcast

The workings of the different shafts are connected by motor-haulage drifts on the even numbered levels. The general location of the orebodies and workings is illustrated by the vertical projection of orebodies of the district as shown in Fig. 1.

The Uncle Sam, Southwest, and Czar workings have many connections to the surface through raises and extensive cracks that were caused by moving ground. These divisions are cool and are ventilated entirely by natural means.

The Holbrook is ventilated partly by natural and partly by artificial ventilation, while the Spray, Gardner, Lowell, and Sacramento are ventilated entirely by mechanical means.

In some of the divisions mentioned above a considerable quantity of air exhausts through shafts of adjoining properties.

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* Mining Engineer, Copper Queen Consolidated Mining Co.

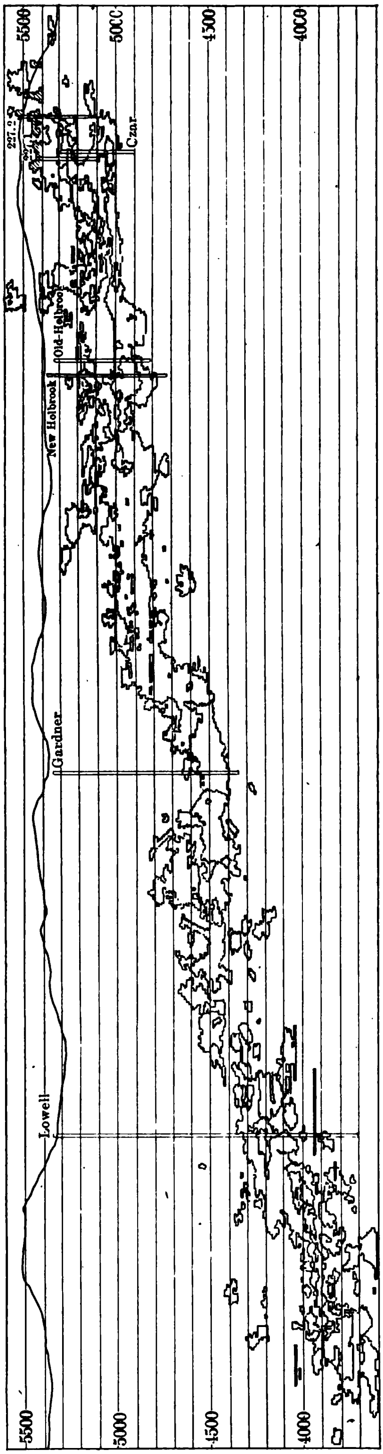

Fig. 1.—Vertical Projection of Orebodies. Plane of projection bears S. 32° 46′ 50″ E.

Conditions Under Natural Ventilation

Several Mine Fires Occurred

During the last few years several mine fires menaced both life and property. The most serious mine fire was that of the Lowell, which occurred during 1911. The district where the fire originated was between the 1,000 and 1,200 ft. levels. The orebody contained a high percentage of sulphur, and the gob a large amount of pyrite; as stoping progressed the friction between the blocks of sulphides increased, due to moving ground. Oxidation of pyrite and timber was also an important factor. The temperature increased until the timbers took fire, which in turn started fire in the sulphide ore. The gases resulting from this fire, such as SO_2, and distillation products of wood, came through the upcast shaft and caused considerable damage, thus putting the shaft out of commission. By reason of broken ground, the fire could not be sealed off at that time, which made necessary the repair of 800 ft. of the shaft by men using oxygen helmets.

Several fires also occurred at different times in some of the other divisions. They were particularly dangerous under conditions of natural ventilation as gases and smoke usually entered the workings. One of the reasons, therefore, in planning a system of mechanical ventilation was to provide a better means of control immediately after a fire started.

General Working Conditions Investigated

During 1912, the company decided to make an investigation of the general working conditions of the Gardner division. By reason of the scarcity of connections with other mines this division contained the poorest working conditions and presented the greatest problem for ventilation. Readings were taken on natural air currents as to their velocity, volume, direction, temperature, relative humidity, CO_2 content, etc. This indicated that even though the entire mine seemed to be hot the average temperature was not high, varying from 75 to 100° F. The relative humidity, however, was exceptionally high everywhere in the mine, ranging from 90 to 100 per cent.

The CO_2 was under 1 per cent. in the general run of working places and from 1 to 3.5 per cent. in exceptionally remote workings. The velocity and volume of the air currents were very low. Moisture was deposited on timbers and the sides of drifts, making the mine damp; a species of fungi covered a large percentage of the timbers. About 225 men worked in this division on each shift, and from the standpoint of efficiency the quantity of fresh air per man per minute was not all that was desired. Men in hot stopes had to be changed every two or three days on account of the heat, and in this way by the time a man was familiar with the workings of his stope he was put into another place. Consequently, all the men who had fairly good working conditions had to share the ordeal from time to time, by working in these undesirable places.

It was the custom when conditions became particularly bad to drive new air connections. In a few instances they proved beneficial, but in the large majority of cases the quantity of air which passed one way or the other was so small that the effect was negligible.

THE RELATIVE MERITS OF PRESSURE AND EXHAUST SYSTEMS CONSIDERED

At first, plans were formulated and data obtained, as described under *Planning the System of Mechanical Ventilation,* for both the pressure and exhaust systems. After the merits of each were carefully considered, the pressure system of ventilation was adopted. According to this system, all mine fans were placed underground on the deepest working levels of the different divisions so as to draw the air down the intake shafts, force it through the workings and stopes, and exhaust it to the surface through upcast shafts.

Four Reasons for Adopting Pressure System

The reasons for adopting the pressure system instead of the exhaust are as follows:

1. *More Practical.*—Since all the deep shafts of the Copper Queen are in operation and a large amount of work is done around each, as for example, men going on and coming off shift, handling timber, tools, and supplies, etc., it would not be practical to have an exhaust fan on the surface near the collar of the shaft.

In the Gardner, a subway below the collar of the shaft would be out of the question. Consequently, a large air-tight chamber, inclosing nearly the entire head frame, would have to be built if a suction fan were used. This arrangement would be expensive and unsatisfactory.

2. *Better Control of Mine Fires.*—The pressure system is the more advantageous in case of mine fires. With this arrangement, the entire workings are under pressure and the gas or smoke from a mine fire may be coursed to exit through neighboring shafts that are not in operation. Since the air is under considerable pressure at the bottom and sides of the present fire districts and each of the outlets over them is provided with regulator doors, the quantity of air passing through the fire zone can be increased or diminished at will and the fire consequently can be brought under control more readily than with the exhaust system.

3. *Greater Efficiency.*—Since the Spray, Gardner, Dallas, and Sacramento shafts are natural downcasts, the mine fans were placed underground so as to take advantage of these circumstances and multiply the volume of the natural downcast from five to ten times. The pressure system, therefore, has the advantage of the additional natural ventilation, which evidently would be lost with the other system; and more power would be required for the same quantity of air if the natural downcasts were changed to upcasts by suction fans.

4. *Greater Safety.*—Experience in the Lowell mine fire showed that in the event of the connections being closed between the fire and the suction fan, there is a possibility of smoke and gases entering the workings below and endangering the men. In the adopted system, the maximum air pressure is below the fire district, which makes it impossible for gases to go down, even if bulkheads of a sealed fire district should be opened suddenly by caving ground.

At present a churn-drill hole is being put down to a depth of 1,100 ft. over the old Lowell fire stopes. When this is completed water will be run in to cool the district. No trouble is anticipated since this was thoroughly tried at the Holbrook, where a large amount of water was turned into churn-drill holes and no smoke or gases were taken down against the pressure of the mine fan.

EQUIPMENT AND GENERAL ARRANGEMENT FOR MECHANICAL VENTILATION

The following pages give an outline of the general mechanical ventilating system which was adopted at the Copper Queen mine.

Combined Natural and Mechanical System in Holbrook and Gardner Divisions

Because of the many openings to the surface, the natural ventilation is satisfactory from the surface to the 400-ft. level in the Holbrook division. Below this level mechanical ventilation is used, as indicated in Fig. 2. There is a 20-in. blower on the 600-ft. level, which delivers 8,000 cu. ft. of air per minute. Since very little stoping is done between the 400- and 600-ft. levels, the blower is operated principally to keep a pressure around the old fire district to prevent any gases which may escape from entering the workings.

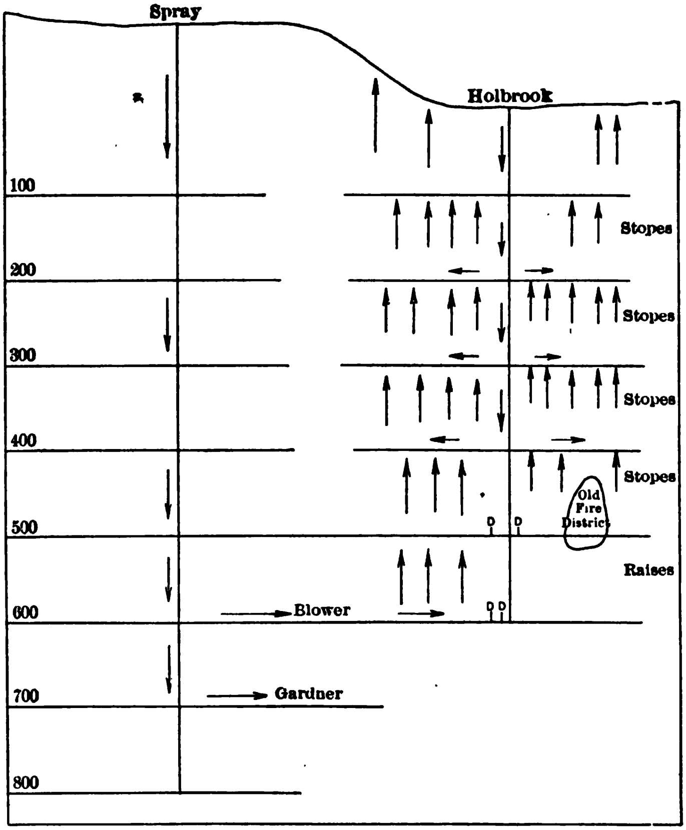

Fig. 2.—Natural and Mechanical Ventilation of Holbrook Division.

The outlines of the natural and mechanical ventilating system of the Gardner division are shown in Figs. 3 and 4. The arrows indicate the direction of some air courses and *DD* represents a few double doors. On the 900-ft. level, there is a Sturtevant mine fan delivering 72,000 cu.

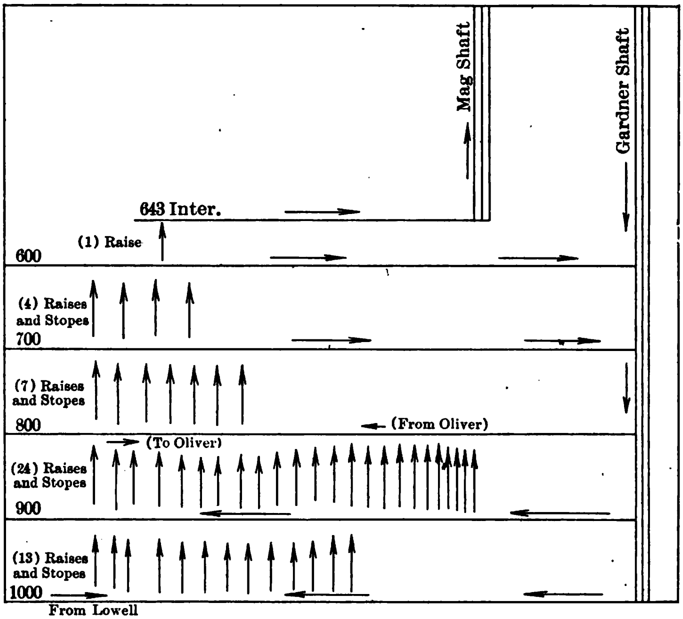

FIG. 3.—NATURAL VENTILATION SYSTEM OF GARDNER MINE.

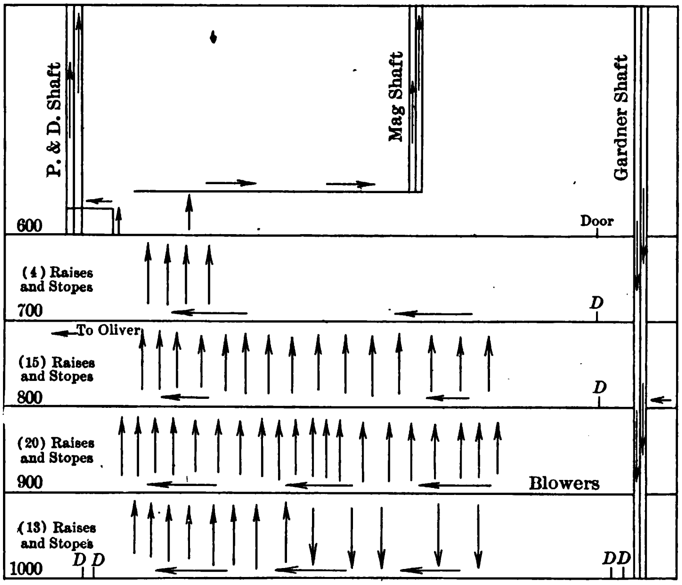

FIG. 4.—MECHANICAL VENTILATION SYSTEM OF GARDNER MINE.

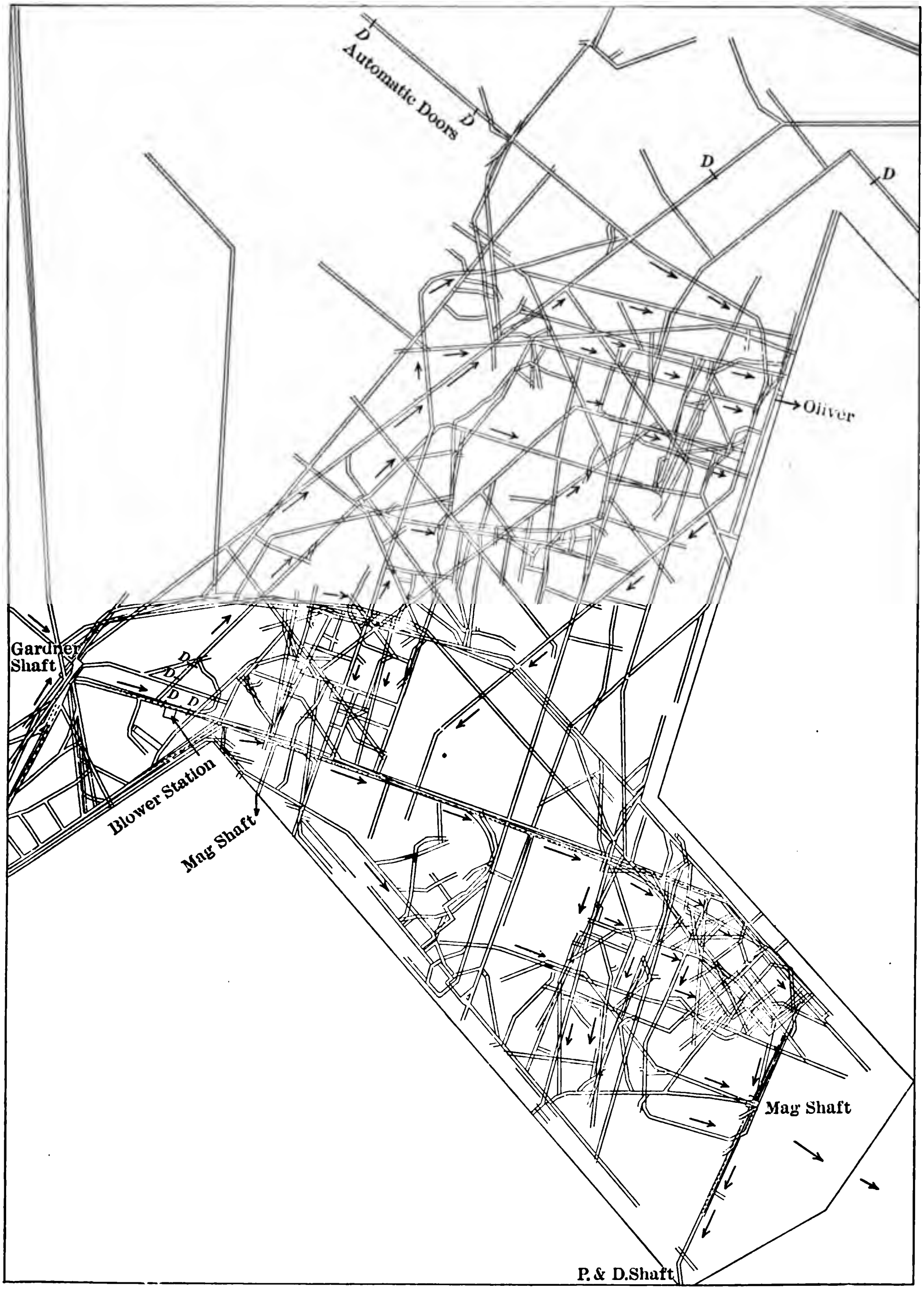

FIG. 5.—ARROWS INDICATE DIRECTION OF SOME IMPORTANT AIR CURRENTS.

ft. of air per minute against 1⅝ in. of water pressure. About 32 h.p. is being used. The combined level map (Fig. 5) illustrates the new connections which had to be driven, also the various directions of air courses when superimposed on one plane. There are 22 hand doors and two large doors on motor tracks, called automatic doors.

The mine fan draws the air down the Gardner and Spray shafts and forces the total quantity into the workings on the 900 level. Part of this air goes down to the 1,000 ft. and later comes up to the 900 ft., from whence all the air passes through stopes to the 800-, 700- and 600-ft., levels, and exhausts through abandoned workings and upcast shafts. On the 1,000-ft. level the pressure is built up by the use of automatic doors. These doors not only hold the pressure, but also allow the air to circulate freely on the 1,000-ft. level. The usual speed of motor trains passing through these doors is about 7 miles an hour. The doors are 350 ft. apart. The arrangement for automatically opening the door is shown in Fig. 6.

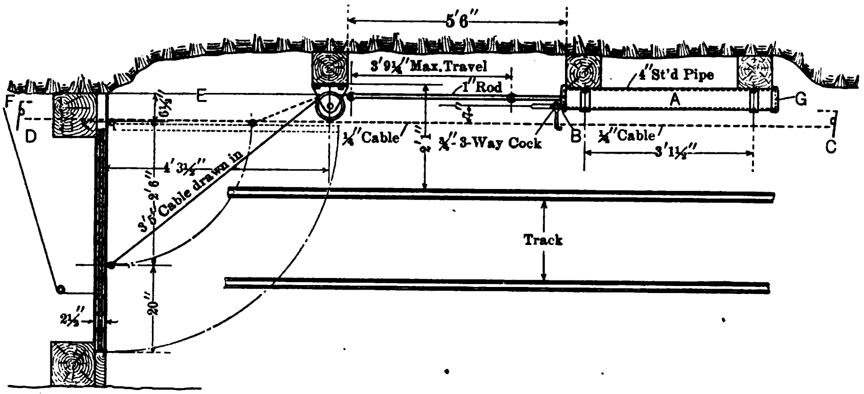

FIG. 6.—PLAN SHOWING AUTOMATIC DOOR ARRANGEMENT.

Operation of the Automatic Door.—When the motor comes in the direction of *C* to *D*, the motorman reaches out and moves the lever *C* in the direction the motor is going. This opens the 3-way valve *B*, admits compressed air into the cylinder *A* and opens the door. When the train has passed through the door the motorman throws the lever *D*, which releases the compressed air and the door closes, assisted by the weights *E* and *F*. The valve *G* controls the speed of opening and closing of the door. The wires between *C* and *D* are crossed so that the levers are always moved in the direction the motor is going in opening and closing the door. The distance between lever *C* and the door is 70 ft., which allows ample time for the door to open when the motor is coming at an ordinary rate. The distance between *D* and the door is 175 ft. This makes it possible for a trip of cars to be in front of the motor on the return and at the same time allow the door to be opened. The cost of upkeep is small and the entire equipment only requires the amount of attention usually given to any machine.

The automatic doors are used on motor tracks only, while small hand doors are used in all ordinary drifts. The standard hand door is 6 ft. 6 in. high, 3 ft. 9 in. wide and 2¼ in. thick, made up of 1-in. boards with paper between the two thicknesses. The posts of the door frame are 6 by 10 in. by 6 ft. 6 in. and the cap 6 by 10 in. by 4 ft. 3 in. The motor tracks are mounted on a timber sill set in concrete. The space between the bottom of the door and the sill is filled with concrete, except for clearance spaces for the flanges of the car wheels. The spaces between the door frame and the walls and roof are likewise filled with concrete. The automatic door is 6 ft. 11 in. high by 4 ft. 4 in. wide made of 1 by 12 in. boards faced on both sides with ¾ by 2 in. flooring.

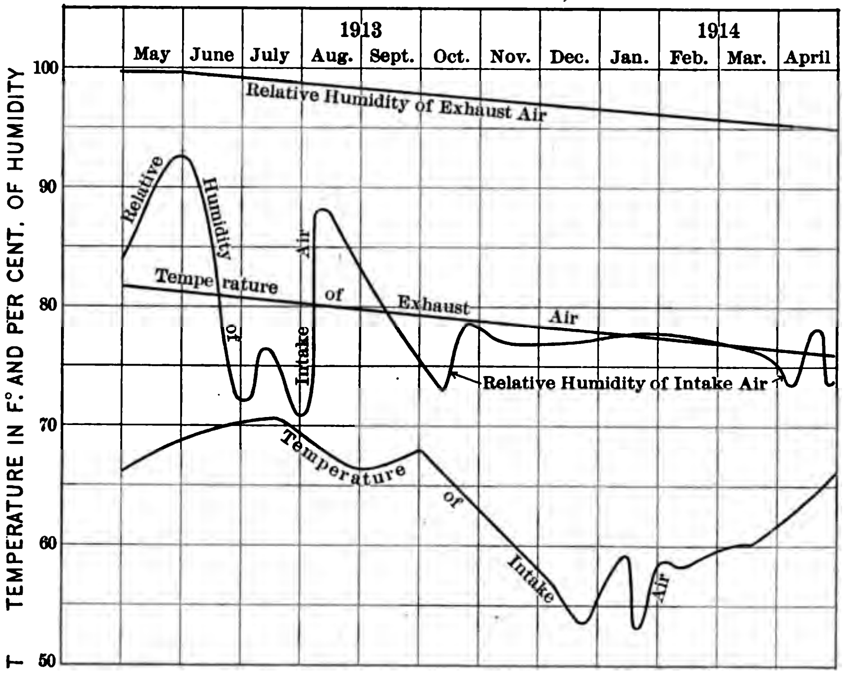

Fig. 7.—Chart of Temperature and Relative Humidity of Intake and Exhaust Air Before and After New Ventilation System was in Operation.

Air Jets Used in Hot Workings.—Formerly small electric blowers were used to ventilate hot raises and stopes. These poorly ventilated places were usually some distance from the shaft, which required a considerable investment and long electric cables. Instead of using these blowers an air jet was designed and tried for workings of this type. The jet had eight $\frac{1}{16}$-in. holes and was placed in the center of a 14-in. funnel which was connected to a 10-in. pipe. About 800 cu. ft. per minute was delivered at the end of 50 ft. of pipe of this kind. The cost of compressed air is comparatively small and is less than one-third the cost of electrical power, when used in these particular instances. Of course the jet is only used until the connection is made. After connections are made considerable quantities of air pass through these drifts and raises from the large blower, and the ventilation is generally satisfactory.

New System Lowered Temperature and Relative Humidity

The chart, Fig. 7, illustrates the decrease in temperature and relative humidity of the intake and return air before and after the blowers were started. On account of the large amount of moisture in the ground and also the wet conditions of moist drifts and workings, it took about four months to dry the timbers and working places after the new ventilating system was in operation.

Velocity and Volume of Air Increased

The blowers were started June 1, 1913, resulting in an increase in velocity of the intake air from 65 ft. per minute on June 1, to 1,000 on June 30 and 1,160 on July 31. The volume also increased from 10,000 cu. ft. per minute on June 1, to over 60,000 on June 30 and 70,000 on July 31. The reduction in the temperature and humidity was not so marked as the increase in velocity and volume. The division in general, however, seems much cooler and the environment is comfortable.

Lowell and Sacramento Divisions Ventilated Jointly

The natural and mechanical ventilation systems of these divisions are outlined in Figs. 8 and 9. There is a Sirocco blower (capacity 35,000 cu. ft.) on the 1,400-ft. level of the Dallas shaft, forcing the air towards the Lowell against a $1\frac{3}{8}$-in. water pressure and another blower on the 1,600-ft. Sacramento (capacity 45,000 cu. ft.) operating against a $1\frac{1}{2}$-in. water gauge. The reason for the higher pressure in the Sacramento is to have the air from that level join the Dallas air on the 1,400-ft. level and from there rise through the workings to the 900-ft. level of the Lowell, and exhaust through the two shafts as indicated in Fig. 9.

Part of Fire District being Cooled.—An example of the effect of large volumes of air passing through a district, part of which is on fire, may be obtained from the 13-10-10 stope. Here the temperature has been reduced from 87 to 83° F. To accomplish this result, 15,000 cu. ft. of air per minute was driven through this stope from Sept. 1, 1914, to Feb.

1, 1915, a period of five months. Before this blower was started the humidity was about 98 per cent. and the velocity of the air was scarcely noticeable. After the 15,000 cu. ft. of air per minute was driven through the stope, even though the temperature showed a very little difference, the humidity dropped to about 80 per cent. and the velocity increased to about 400 ft. per minute. As a result the stope seems cool and the men can do efficient mining. This small reduction in temperature also shows that it requires a large quantity of air circulating a long time in a fire district to cool the country rock.

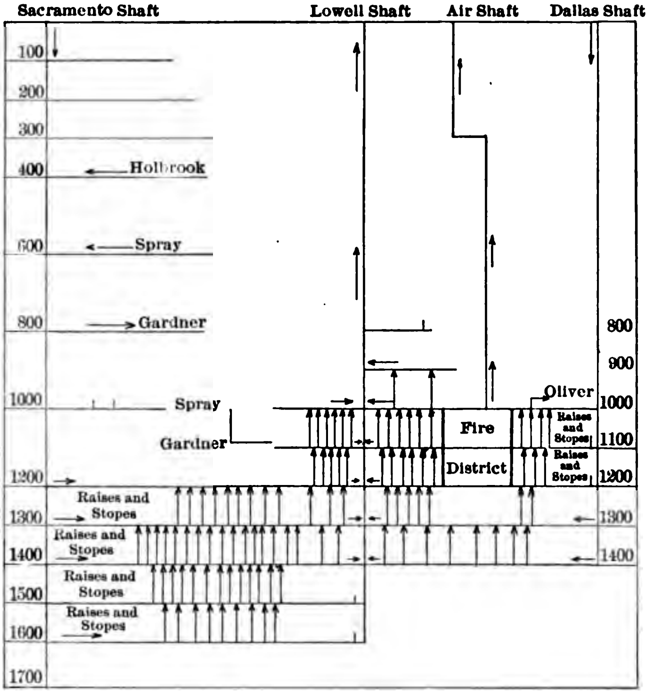

FIG. 8.—OUTLINE OF NATURAL VENTILATION SYSTEM OF LOWELL AND SACRAMENTO MINES.

The velocity and volume of air have been increased through all the stopes and workings and although the temperature throughout the entire mine has not been lowered much, nevertheless the workings seem much cooler on account of the increased velocity and volume and decrease in humidity.

DATA COLLECTED BEFORE MECHANICAL VENTILATION SYSTEM PLANNED

In order to plan the general system of mechanical ventilation of the Copper Queen, as outlined above, the following methods of obtaining data for a working basis were pursued. It may also be stated here that these methods would be the same for either pressure or exhaust systems.

Instruments for Determinations.—The following instruments were used in determining the temperature, relative humidity, velocity and volume of mine air: The anemometer, water gauge, hygrometer, and barometer.

Recording of Data.—Data were readily obtained in the form of daily reports.

Barometric readings, when taken occasionally in important places, were satisfactory for all practical purposes, while readings with the anemometer and hygrometer were made daily for several months.

The following is an example of the daily report blank used at the Copper Queen:

Ventilation Report

Shaft______Level______Dry Bulb______Wet Bulb______Humidity______Bar______Date______

Location	Date	Time	Area	Veloc.	Cu. Ft.	Bar	Dry Bulb	Wet Bulb	Relative Humidity	CO_2	Air Current
Average											

Two Standards of Measurement

There are two standards by which the degree of ventilation in a mine may be measured: (a) a quantity standard—that is, the quantity of pure air entering the mine per man, per minute; or, (b) the quality standard—this being determined by the amount of impurity present. These standards are entirely different and each one has its particular application.

The quantity standard is less expensive, more practicable, and is the method most generally used in both coal and metal mines. The quantity standard was used in all drifts, raises, and places where there are air currents that can be detected and measured with the anemometer. This method is also the more important because the increase of velocity of air materially affects the efficiency of the men.

The quality standard is very expensive and its application is only necessary in exceptional cases. It has only a remote bearing on the efficiency of the men, but occasionally it is found advisable to have accurate determinations made on several samples of mine air.

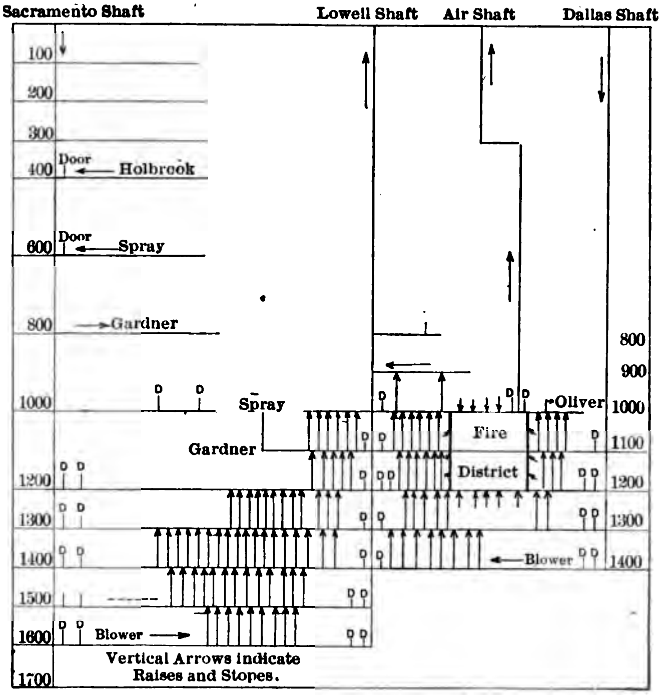

FIG. 9.—OUTLINE OF MECHANICAL VENTILATION SYSTEM OF LOWELL AND SACRAMENTO MINES.

Averages Plotted on the Mine Map

The data secured as explained in the paragraph on "Recording of Data" were averaged every week. Blueprints of the different levels of the mine were made and the monthly averages written on them in their proper places. The direction of air currents was also indicated by arrows, especially where readings were made with the anemometer.

Amount of Air Necessary for Each District Calculated

In order to calculate the amount of air necessary for each district, it was necessary to take into account the following:

1. Number of men and animals in each district.
2. The production of CO_2, or other gases, as shown by the chemical analyses.
3. Relative humidity.
4. Temperature.
5. Amount of explosives used.
6. The distance from currents of good air.
7. The number of lights.
8. Air leakage.
9. Friction of the air currents.
10. Number of splits of the air current.
11. Method of distribution.

Natural Ventilation Utilized Where Possible

In order to get the most economical system, it was necessary to utilize all possible natural ventilation and secure the proper distribution of the air. The latter is extremely important because the efficiency of the ventilating system depends largely on the proper distribution of the air currents. The arrangement of air currents by mechanical means was designed to coincide with that of natural ventilation in nearly all cases.

Natural Ventilation Supplemented by Mechanical Means

In the divisions where natural ventilation was not sufficient, the natural air currents were strengthened by mechanical, or artificial means, by installing large mine fans.

A Good Working Atmosphere Dependent on Several Factors

There is a diversity of opinion among authorities on what constitutes "a good working atmosphere," and to what extent it will be economical to ventilate a mine, considering the cost of installation, maintenance, and power. The quantity of air necessary for the ventilation of one mine would be inadequate for the ventilation of another. The same can be said regarding the quantity of air per man per minute, consequently every mine is an individual problem, and while generalities are made, to get the greatest efficiency of a ventilating system, a detailed study of the mine must first be made, and the new ventilating system adapted to meet the particular needs of each mine.

In a number of States the quantity of air in coal mines required by law is usually specified, as, for example: The Anthracite Mine Law of Pennsylvania specifies a minimum quantity of 200 cu. ft. per man, per minute. One of the modifying clauses of the above law stipulates that the amount of air in circulation should be sufficient "to dilute, render harmless, and sweep away, smoke and noxious or dangerous gases."

Even in coal mines, therefore, where the problems of ventilation have not the intricacies found in metal mines, there is no fixed standard upon which to base figures for "a good working atmosphere" or an efficient ventilating system.

Experience in ventilation of the Copper Queen mine has shown thus far that in poorly ventilated areas, comfort, efficiency, and economy are obtained by increasing the natural ventilation with mechanical means until the following conditions are obtained: Temperature, 77° F.; relative humidity, under 85 per cent.; velocity of air in working places at least 100 ft. per min.; volume of air in working places at least 250 cu. ft. per man, per minute. This may be termed a good working atmosphere and is the present standard for this camp.

In general, if the above velocity and volume are maintained in any working place it will not be necessary to make chemical analyses, because the percentages of CO_2, CO, NO_2, and SO_2 gases, etc., as well as the temperature and humidity will be low and require no attention, except in a few instances.

Increased Velocity of Air Improves Working Conditions

Formerly there were several examples of stopes in the Copper Queen mine which were similar to the following:

In No. 8–13–5 stope of the Gardner division, the readings in working places for May 28, 1913, were

Velocity	10 ft. per minute (approx.)
Volume	10 cu. ft. per min. (approx.)
Temperature	84° F.
Humidity	96 per cent.

A small 5-h.p. blower was forcing air into the stope. The miners were uncomfortable in this stope and complained about the excessive heat.

On Oct. 1, 1913, after the new ventilating system was installed, the following readings were made in the same stope:

Velocity	100 ft. per minute
Volume	500 cu. ft. per min.
Temperature	84° F.
Humidity	90 per cent.

The same miners mentioned above happened to be working in this stope on Oct. 1, 1913, and they remarked "the stope is not near as hot as it used to be and we can work much better." The men did not sweat excessively in this atmosphere, and the greater comfort experienced by them was due to the *increase in velocity*, which also means an increase in volume, with lower humidity, and not to the temperature, as it was the same in both cases.

When miners work in hot mines, drifts, and stopes, they generally neglect their personal safety. It was a frequent occurrence for the shift boss to enter a working place and after asking miners to bar down the back to discover that they had been working under loose boulders for several hours. It is evident, therefore, and proved by the rate of accidents, that men are on the alert and look out for personal safety far more in a comfortable atmosphere than in hot, damp, and disagreeable places.

In regard to efficiency, the tonnage per man shift has increased with the increase in the velocity and volume of air resulting from the installation of large mine fans. The greater efficiency of the men may be partly assigned to better organization and cooperation and partly to the psychological effect resulting from good ventilation. For instance, in the language of the average miner, he "tries to make a showing every day." In a comfortable atmosphere, he will take an interest in his work, and go about it in a more contented spirit than he could possibly do under hot, unfavorable, unhealthy, and depressing conditions.

A special effort, therefore, is constantly being made at the Copper Queen to increase the velocity of the air in all the raises, drifts, and stopes, and keep the ventilating system up-to-date with the newly developed workings.

CHANGES AFTER MECHANICAL VENTILATING SYSTEM WAS COMPLETED

Development Work and Ventilation System Planned Together

In general, the mine department makes an effort to plan development work in line with the ventilating system, as well as for prospecting purposes. The drifts are usually driven about 5 by 8 ft. and raises are run two compartments each, 50 in. in the clear. Recently standard raises have been adopted throughout the mine, and the timber compartments as well as manways have also been standardized. Rails are put over timber compartments to allow free passage of air (see Fig. 10) and manways have been made larger to permit more air to pass through them and at the same time to make it easier for the men to go up and down. Also when an orebody is found it is properly developed and prospected before stoping is commenced. In this way ventilation is really first established, while the orebody is being brought into a condition for stoping operations. This is a great help in the ventilation of stopes.

Square-Set and Shrinkage Stopes Ventilated Easiest

Recently other methods of stoping have been giving good results besides the original square set. Shrinkage, cut-and-fill, and top-slice stopes have progressed far enough to show that each method has its particular application to certain kinds of ground. In ventilating stopes under these systems, probably the most easily ventilated are the square-set and shrinkage stopes. Next to these are the cut-and-fill, and the most difficult the top-slice stopes. However, in all cases at present where the top slice is being used, separate drifts and raises were driven to facilitate the handling of timber and tools and at the same time to give connections for ventilating purposes. Consequently, the top-slice stopes have about the same temperature and humidity as found in the square-set sections.

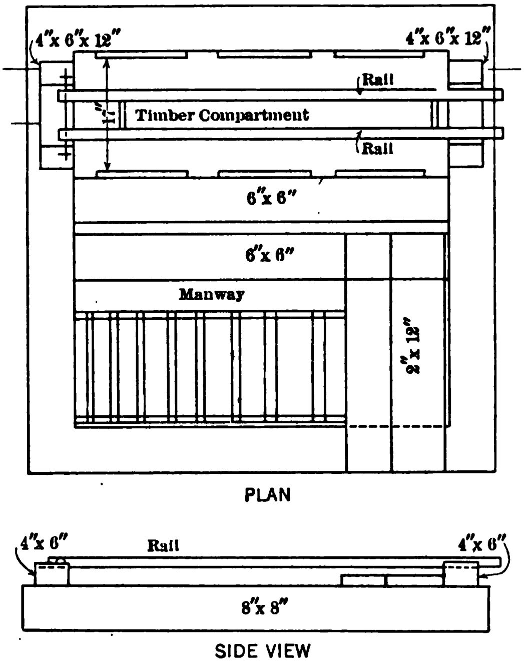

FIG. 10.—METHOD OF COVERING TIMBER COMPARTMENTS.

ADVANTAGES OF NEW VENTILATING SYSTEM

1. A smaller quantity of compressed air is used for ventilation.
2. All possible natural ventilation is utilized and the general direction of air currents is the same for both natural and mechanical ventilation.
3. There is a decrease in temperature and a lower percentage of humidity and CO_2 throughout the mine, which means healthier working conditions.
4. A saving of eight 5-h.p. blowers was made, as these have been removed from the mine.
5. The five lower divisions are under pressure, which partly acts as

a check in preventing some mine fires from crossing the Copper Queen side lines, as for example, the proximity of one mine fire near the Gardner workings.

6. Less decay of mine timbers.

7. The volume of air has been increased from 60 cu. ft. of air per man per minute, to about 300 cu. ft.

8. Formerly a large amount of moisture kept the timbers and working places in a damp condition and a species of fungi also covered most of the timbers. Since the mine fans were started, a large amount of water is removed daily from each division by the air currents so that the mine is comparatively dry and the fungi have disappeared.

9. The increase in velocity and volume of air resulted in the greater efficiency of the men.

10. The new ventilating system is advantageous to the mines adjacent to the Copper Queen mine; there is a great deal of broken and stoped ground along side lines permitting a large amount of air to escape through these avenues and ventilate stopes beyond the Copper Queen boundary.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should be in the form of a new paper.

The Formation of the Oxidized Ores of Zinc from the Sulphide

BY YINCHANG TSENSHAN WANG, NEW YORK, N. Y.

(San Francisco Meeting, September, 1915)

CONTENTS

I. Introduction:
  1. Subject and Scope.
  2. Acknowledgments.
  3. Review of Previous Work.
    Field Data
    Experimental Data
  4. Plan of Present Investigation
    Method of General Procedure
II. Experimental Work:
  A. The Solution of Sphalerite.
    1. The Solvent Effect of Iron Sulphates upon Sphalerite.
    2. The Solubility of Sphalerite in Salt Solutions of Silver, Copper and Lead.
  B. Formation of Zinc Carbonates.
    3. The Reactions of Carbonates and Bicarbonates With Solutions of Zinc Salts.
    4. The Reaction of Zinc Sulphate Solution With Limestones.
  C. Peculiar Conditions Affecting the Formation of Zinc Silicate (calamine) and Zinc Carbonate (smithsonite).
    5a. The Effect of Carbon Dioxide Upon the Basic Silicate and the Carbonates of Zinc in Water, and Its Bearing Upon the Conditions of Their Precipitation.
    5b. The Reactions of Solutions of Soluble Silicate With Calamine and Smithsonite in Solution in Presence of Carbon Dioxide.
    6. The Reactions of a Mixture of Hydroxide and Silicate of Sodium with a Dilute Solution of Zinc Sulphate.
  D. High-Temperature Methods.
    7a. The Influence of Temperature on the Formation of Calamine.
    7b. Possible Effect of Heat on Calamine and Smithsonite.
  Summary of the Experimental Work.
III. Petrographic Work:
  1. The Occurrence of Sphalerite in Limestones or in Cherts.
    Summary of the Blende in Limestones and in Cherts.
  2. The Partially Altered Sphalerite and the Oxidation Products.
    a. Smithsonite with Blende.
    b. Calamine with Blende.
    c. Smithsonite and Calamine.
    d. Summary on the Partially Altered Sphalerite and Its Oxidation Products.
IV. Conclusions.
V. Photomicrographs and Photographs with Their Explanations.

I. INTRODUCTION

1. *Subject and Scope*

The formation of the oxidized ores of zinc has been recognized as a subject of great importance in economic geology. For many years these ores have been much sought, partly because they are more desirable for metallurgical treatment and partly because they are apt to be freer from gangue than is the sulphide of zinc. Up to very recent years the oxidized ores were the principal source of the metal and at present they still furnish a large portion of the commercial production.

The oxidized ores of zinc include six minerals, as shown in the following table:

Mineralogical Name	Chemical Name	Chemical Composition	Constituents, Per Cent.			
			Zn	SiO_2	Fe	Mn
Calamine....	Basic silicate of zinc	$2ZnO.SiO.H_2O$ or $Zn(OH)_2.ZnSiO_3$.......	54.20	25.0	......	......
Smithsonite.	Zinc carbonate......	$ZnCO_3$.............	52.00	......	......	......
Hydrozincite	Hydro-carbonate of zinc...............	$ZnCO_3.2Zn(OH)_2$ ...	......	......	......	......
Willemite...	Anhydrous silicate of zinc..............	$2ZnO.SiO_2$ or $ZnO.ZnSiO_3$...........	58.50	27.0	......	......
Zincite......	Zinc oxide..........	ZnO...............	80.30	......	......	......
Franklinite..	Iron-manganese-zinc oxide............	$(FeZnMn)O(FeMn)_2O_3$.............	12.00	......	42.0	15.0

Only the first three, calamine, smithsonite and hydrozincite have been considered in this investigation. Wherever the terms "oxidized ores" and "oxidation products" are used in this paper they are, unless otherwise indicated, strictly limited to these three ore-minerals of zinc.

These oxidation products are universally believed to be derived from the sulphide of zinc, as their original mineral. The chemical reactions of the change, however, are not fully understood and have not, heretofore, been studied experimentally in the light of physico-chemical principles. The influence of oxidation by surface waters and of other metallic compounds which are associated with the blende, is only inferred from field relations. At the suggestion of Dr. J. F. Kemp, the writer undertook a series of experiments with the hope of clearing up some important points in connection with the chemistry of the precipitation of the three oxidized ores and in particular, of establishing the peculiar set of conditions under which the antecedent oxidation of the blende has taken place.

The principal part of the experiments is, in accordance with the geological evidence, limited to wet methods, and deals with a single substance, or a group of substances in solution, which are capable of effecting the solution of the blende or of converting its oxidation product into a carbonate or a basic silicate, as the case may be.

2. *Acknowledgments*

The writer takes great pleasure in acknowledging his indebtedness to his professors and friends for aid received in the preparation of this paper. First among them is Professor J. F. Kemp, of Columbia University, under whom this work was done. Throughout the entire laboratory work, and especially on the experimental side, he has manifested great interest in the results as they appeared. Whatever merits this paper may have are largely due to his valuable assistance and advice. For many useful criticisms relative to the petrographic feature of the work, the writer is indebted to Professor C. P. Berkey, of the same University. The writer's thanks are also due to Professor T. C. Chamberlin, of the University of Chicago, for inspiration and encouragement through correspondence and otherwise during the execution of this research, and to Dr. R. T. Chamberlin for many helpful suggestions. To Professor R. J. Colony of Cooper Union of New York City, the writer is indebted for checking the results of optical examinations of the crushed fragments. Acknowledgments to other friends have been indicated in the course of the following pages.

3. *Review of Previous Work*

Field Data.—Field data concerning the paragenesis of the oxidized ores of zinc are abundant. The most suggestive contribution to the literature of the subject is by Dr. R. Beck.[1] In the discussion of the zinc deposits at Raibl in Carinthia, Austria, he regards the oxidized ores of zinc in that region as the results of extensive metasomatism, and says that the limestone, the inclosing rock of the ores, "is directly replaced by calamine and the structure of the original rock is retained by the calamine and carbonate of zinc." Our general experience shows that calamine and smithsonite are derived from blende. Therefore, the solutions which replaced the soluble limestone presumably obtained their salts by the oxidation of sphalerite and the process was essentially one of secondary enrichment. That is to say, the oxidation and solution of the sphalerite must equally depend: (1) on the action of the oxidation products of pyrite or marcasite on the sphalerite in presence of water (these iron sulphides in nature are invariably associated with zinc sulphide); (2) on the direct oxidation of sphalerite as a result of electrolytic action[2]

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[1] R. Beck. *Lehre von den Erzlagerstätten,* Band 2, p. 596 (1901).

[2] V. H. Gottschalk, and H. A. Buehler. Oxidation of Sulphides, *Economic Geology,* vol. vii, p. 28 (1912).

TALE I.—*Analyses of Mine Waters and Well Waters in Parts per Million*

	1	2	3	4	5	6	7	8	9	10
S*........	184.2	1886.0	2033.4	3200.3	—	221.4	429.0	154.3	29.7	161.0
Cl........	0.0	0.0	0.0	0.0	0.0	0.0	12.8	67.45	3.5	—
Free H_2SO_4...	0.0	220.5	39.0	1568.0	333.2	0.0	0.0	0.0	0.0	0.0
Combined H_2SO_4...	0.0	0.0	6060.9	8032.9	0.0	0.0	0.0	0.0	0.0	0.0
Metallic Fe	0.0	670.0	362.0	523.5	465.0	—	0.0	—	1.0	15.5
Fe........	0.0	490.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
Fe........	0.0	180.0	293.0	1091.5	—	0.0	0.0	3.0	—	—
CaO......	354.1	—	700.5	—	—	—	501.0	333.9	—	—
MgO......	31.4	—	0.0	—	—	—	—?	108.6	—	—
Pb........	0.0	0.0	0.30	0.0	0.0	0.0	0.0	0.0	1.3	0.0
Metallic Zn.......	20.0	1432.5	80.50	2250.0	910.0	20.9	257.0	1.7	7.3	70.0
Acidity						Alkali	ne			

	11	12	13		14	15	16	17
KCl.......	9.5	6.1	0.0	SO_4	233.4	222.4	215.9	104.0
K_2SO_4.....	0.0	0.0	0.0	SiO_2	6.8	8.0	2.1	18.0
NaCl......	0.0	1.3	4.5	Al_2O_3	2.8	0.7	2.8	0.0
Na_2SO_4....	130.3	38.5	30.8	FeO&Fe_2O_3	6.7	9.0	6.8	9.5
$Al_2(SO_4)_3$..	68.3	73.6	3.3	MnO	2.7	5.9	8.3	0.0
$CaSO_4$....	1528.0	924.6	371.5	ZnO	23.5	5.9	3.5	11.1
$Ca(HCO_3)_2$	0.0	0.0	0.0	Cl	trace	trace	trace	12.4
$MgSO_4$....	trace	192.6	60.4	CaO	148.5	101.4	127.0	65.0
$FeSO_4$.....	611.9	387.7	2.6	MgO	28.4	24.7	21.7	24.9
$PbSO_4$.....	0.04	0.0		Insoluble residue....	17.5	15.8	21.0	0.0
$ZnSO_4$.....	586.2	851.7						
SiO_2.......	23.2	23.2	7.4					
H_2SO_4.....	170.9	251.7	0.0					
CO_2(free)..	12.0	87.0	19.7					

1. Oronogo Circle mine, Oronogo, Mo.
2. Duenweg mine, Joplin district, Mo.
3. Continental mine, Joplin district, Mo.
4. Mary mine, Joplin district, Mo.
5. Diamond Jack mine, Carterville, Mo.
6. 9th Shaft, Oronogo Circle mine, Oronogo, Mo.
7. Well on Centre Creek.
8. Well on Main Street, Joplin, Mo.
9. Centre Creek water.
10. Centre Creek water.
11. Arkansas mine, Joplin district, Mo.
12. Victor mine, Joplin district, Mo.
13. O. K. mine, Joplin district, Mo.

14–17. Mine waters of Rothschonberger Stolen, near Freiberg, Germany.

— = No determination was made of the element.

O = The element was absent.

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* The sulphur in all cases exists as sulphate.

due to the contact of sphalerite and iron sulphides; or (3) on both of these actions.

In addition to this statement of Beck's, we have the evidence afforded by analyses of mine waters and of well waters from zinc districts.

> "While the waters taken from a mine may differ slightly in composition from the ground waters that seeped through the undisturbed deposit, in that they are probably more dilute on account of the freer circulation of the solutions, and are also more highly oxidized on account of greater access to the atmosphere, yet nevertheless their composition approaches fairly closely that of the original ground-waters."[3]

The Table I comprises typical analyses[4] of the waters from zinc mines both in Europe and in America, and of the waters from wells in the zinc district of Joplin, Mo. The figures given are all in parts per million.

On inspection of these data, it will be noted that sulphur in all cases exists as sulphate, indicating as a distinctive feature of these waters, the predominance of the sulphates of zinc, iron, calcium and generally magnesium, over other salts. It has also been noted, according to Mr. Waring,[5] the Chemist of Webb City, Mo., that the iron in the waters freshly drawn from the depths of the Joplin mines, is always present as ferrous sulphate, while the waters from the surface levels frequently contain ferric sulphate. Further evidence has recently been shown by Hodge[6] as to the presence of ferrous salts in the waters at some depth and of ferric near the surface.

According to Allen[7] and Wells,[8] the excess of sulphuric acid formed by the oxidation of the iron sulphides is responsible for the increased solubility of the metallic sulphides in association. This assumption does not seem to be entirely consistent with the experimental facts,[9] or to be generally supported by chemical analysis. Mr. Waring,[10] speaking from his wide experience in the zinc mines of Joplin district, Mo., says that free sulphuric is not always present; in some cases, the waters that are quite rich in zinc sulphate contain no sulphuric acid. On the contrary they gave a strong alkaline reaction to methyl orange, frequently containing considerable amount of calcium bicarbonate. The presence of the latter compound in solution may be a factor in the process of the formation of zinc carbonate and will be discussed experimentally later in this paper.

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[3] N. C. Cooke. *Journal of Geology*, vol. xxi, p. 5 (1913).

[4] For the analyses Nos. 1 to 10, the writer is indebted to W. G. Waring of Webb City, Mo., who has very kindly placed at his disposal the data which for the first time appears in print. For Nos. 11 to 13 inclusive, the writer has quoted from a paper by Messers Buehler and Gottschalk, *Economic Geology*, vol. v, p. 34 (1910); Nos. 14 to 17 from *Lehre von den Erzlagerstätten*, R. Beck, vol. ii, p. 402 (1901).

[5] Private communication.

[6] E. T. Hodge. *Economic Geology*, vol. x, p. 123 (1915).

[7] E. T. Allen. *Economic Geology*, vol. v, p. 387 (1910).

[8] R. C. Wells. *Economic Geology*, vol. v, p. 480 (1910).

[9] Gottschalk and Buehler. *Economic Geology*, vol. vii, p. 27 (1912).

[10] Private communication.

Experimental Data.—In the literature of this subject no experimental data bearing on this problem were found. Those who have dealt with the question in the laboratory have rarely worked under the conditions which commonly obtain in nature; *i.e.*, low temperatures, low pressures, small concentrations, and the combination of all these conditions, presenting a narrow range of possibilities for imitative synthetic studies. H. de Senarmont[11] showed that zinc carbonate can be formed from zinc chloride and calcium carbonate at 150°C. Gorgeu[12] in 1887 claimed that he obtained the anhydrous silicate and the oxide of zinc (willemite and zincite) by fusing a mixture of chemical compounds containing the required constituents. His experiments have been repeated by the writer, but without producing the same results.[13] No attempt has heretofore been made, even using high temperature, to produce synthetically the basic silicate of zinc (calamine).

The investigations of Allen and Creushaw[14] on zinc compounds have no bearing on this problem, since their work was confined to the study of zinc sulphides.

4. *Plan of Present Investigation*

The present work was divided into two parts, experimental work and microscopic studies of thin sections. Experimentally the main purpose was to find: (a) what is the exact nature of the compound or compounds, commonly associated with the sulphide of zinc, which cause it to be oxidized and carried into solution in meteoric waters; (b) what are the most common reagents or compounds which are likely to have caused the percipitation of carbonates or the basic silicate of zinc; (c) what are the peculiar conditions under which these oxidation products may be obtained in the laboratory; (d) to what extent the formation of these oxidation products is limited by ordinary laboratory conditions; and (e) what influence is exerted by temperature. The microscopic study was done chiefly in the hope of obtaining further evidence throwing more light on the problem.

The experimental work here described deals with the following questions:

1. The solvent effect of iron sulphates on sphalerite.
2. The solubility of sphalerite in a salt solution of silver, of copper and of lead.

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[11] Doelter y Asterich. *Handbuch der Mineralchemie*, Band 1, p. 499.

[12] *Bulletin* 10, *Sociétié Française de Mineralogie*, pp. 30 to 39.

[13] See the discussion under "The Influence of Temperature on the Formation of Calamine," Experiment 12.

[14] *American Journal of Science* 4 *series*, vol. xxxiv, pp. 341 to 360 (1912).

3. The reactions of carbonates and bicarbonates with solutions of zinc salts.
4. The reaction of zinc sulphate solution with pure and impure limestones.
5. The effect of carbon dioxide upon the basic silicate and the carbonates of zinc in water, and its bearing upon the conditions of their precipitation.
6. The reactions of a mixture of hydroxide and silicate of sodium with dilute solutions of a zinc salt.
7. The influence of temperature on the formation of calamine, and its possible effects on natural calamine and smithsonite.

On account of the wide range of the subjects involved in the experiments, each series will be described and discussed under its own heading. Unless otherwise indicated, all of these experiments were conducted under ordinary conditions of temperature and pressure. Concentrations of solutions were small, deci-normal in some cases, and fiftieth in others; these concentrations may slightly differ in behavior from those commonly obtained from the ground waters, but the difference is only in the speed, and not in the nature, of the chemical reactions.

The paper also includes a discussion, based on the suggestion of Dr. Kemp[15] as to the origin of the zinc deposits at Franklin Furnace, and Ogdensburg, N. J., of the effect of heat on calamine and smithsonite, with possible changes to willemite and zincite respectively.

Method of General Procedure.—Unless otherwise indicated, the mineral to be tested was first powdered; and only the portion which passed 50- or 100-mesh sieve, as the case might be, was used. Chemically made zinc compounds were also used and are indicated in each case. The amount of reagents in solution, as to their constituents, were definitely known whenever it was deemed necessary. In all cases, the precipitate, the residue if any, and the liquid contents in filtrates, were determined quantitatively. The comments which usually follow the record of the experiment were made in each case in consequence of the laboratory results. In determining the temperatures of formation of the oxidation products (here including willemite and zincite but excluding hydrozincite) both the quantitative analysis and microscopic examination of the crushed fragments were employed.

II. EXPERIMENTAL WORK

A. *The Solution of Sphalerite*

1. *The Solvent Effect of Iron Sulphates upon Sphalerite.*—As already remarked, there is commonly present in the freshly drawn water from the depths of zinc mines, such as those of Joplin, an iron salt in solution,

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[15] *Transactions of New York Academy of Science*, vol. xiii, p. 92.

which, if drawn from some depth, is invariably in the form of ferrous sulphate, and doubtless indicates the oxidation of the blende to some extent at the expense of the reduction of ferric sulphate.

The ferric sulphate is presumably produced by the oxidation of an iron bisulphide (pyrite or marcasite) which occurs not infrequently in the zinc-producing districts. The reaction is ordinarily written:

(*a*) $FeS_2+7O+H_2O \rightarrow FeSO_4+H_2SO_4$; and there is further oxidized:

(*b*) $2FeSO_4 + H_2SO_4 + O \rightarrow Fe_2(SO_4)_3 + H_2O$. This ferric solution may hydrolize to

(*c*) $Fe_2(SO_4)_3 + 3H_2O \rightarrow Fe_2O_3 + 3H_2SO_4$.

Reaction (*c*), however, occurs only in the absence of such minerals as blende. In blende-bearing strata the reaction (*c*) does not occur; at least, no such case has ever been reported. On the contrary, as shown by experimental results recorded below, when this strongly oxidizing agent is held in waters which percolate into blende-bearing rocks, it will react with the sphalerite and allow zinc to pass into solution. The reaction is in accordance with the equation:

$ZnS + Fe_2(SO_4)_3 \rightarrow 2FeSO_4 + ZnSO_4 + S$

In order to determine just what proportion of the ferric sulphate is necessary to oxidize one part of sphalerite, the following experiments were undertaken:

Experiment 1.—A piece of sphalerite obtained from Joplin, Mo.,

	I (a)		I (b)		II	III		IV	
	Actual amounts originally present in grams				Conditions in the filtrate before analysis. Grams.	Filtrate		Residue	
	In solution		In solid						
	Element	Compound	Element	Compound		Grams	Per Cent.	Grams	Per Cent.
S			1.598					0.3055	18.84*
Impurity								0.0132	
Zn			3.262					trace	
Fe^{++}						0.8794	6.304		
Fe^{+++}	1.8636					0.9833	7.049		
SO_4	4.8027					8.8270	63.270		
Zn						3.2550	23.330		
$FeSO_4$					2.3900				
$Fe_2(SO_4)_3$		6.6663			3.5190				
$ZnSO_4$					8.0350				
ZnS				4.860					
Total	6.6663	6.6663	4.860	4.860	13.9447	13.9447	99.953	0.3187	18.84

*This shows that the rest of the sulphur (81.16 per cent.) has been oxidized into sulphate.

and known to contain 0.9972 g. ZnS for every gram of the sample was ground to pass a 50-mesh sieve; 4.873 g. of this material (on account of its impurity, this amount is a little less than $\frac{N}{10}$ of its molecular weight) was digested in 1,000 c.c. ferric sulphate solution, containing 6.663 g. of the salt (approximately $\frac{N}{10}$). The reaction is manifested as soon as the sphalerite is placed in the ferric solution. The mixture was allowed to stand in a 2-liter bottle, whose mouth was closed with a bunsen valve, and also attached by a side glass tube to an air tank from which ordinary air was allowed to pass slowly into the mixture 15 min. at a time, three times a day, for a period of four weeks. The foregoing table shows the analyses of both the residue, which weighed 0.3204 g. and the filtrate.

Experiment 2.—An equal amount, 4.873 g. of lump sphalerite from the same sample was treated in 1,000 c.c. $\frac{N}{10}$ ferric sulphate solution. In the same way as in Experiment 1, ordinary air was allowed to pass slowly into the mixture three times a day, 15 min. at a time, for a period of four weeks. The residue weighed 2.0154. g. The following analyses were obtained from the residue and the filtrate:

	I (a)		I (b)		II	III		IV	
	Actual amounts originally present in grams				Conditions in the filtrate before analysis. Grams	Filtrate		Residue	
	In solution		In solid						
	Element	Compound	Element	Compound		Grams	Per Cent.	Grams	Per Cent.
S			1.598					0.8572	53.23*
Impurity								0.0136	
Zn			3.262					1.1490	
Fe^{++}									
Fe^{+++}	1.8636					1.1210	10.28		
SO_4	4.8027					0.7413	6.795		
Zn						6.9380	63.59		
$FeSO_4$					3.047	2.1120	19.36		
$Fe2(SO_4)_3$		6.6663			2.6523				
$ZnSO_4$					5.213				
ZnS				4.860					
Total	6.6663	6.6663	4.860	4.860	10.9123	10.9123	100.025	2.0198	53.23

* This shows that the rest of the sulphur (46.77 per cent.) has been oxidized into sulphate.

In both filtrates the iron and sulphate contents were determined with a known strength of $K_2Cr_2O_7$ and $BaCl_2$ solutions respectively. The amount of the sulphates of ferrous and ferric, together with that of zinc, in the filtrate was recalculated and recorded in column II. These analyses also show an excess of sulphur in the residue, indicating the presence of sulphuric acid, which is sometimes present with $FeSO_4$ and $ZnSO_4$ in the waters of zinc mines.[16]

According to the above results, there was practically no zinc remaining in the residue in experiment 1, and the amount of ferric (47.21 per cent.) reduced to ferrous, as a result of the oxidation of ZnS, in the first case is 12.96 per cent. less than in experiment 2, in which, in spite of the presence of more ferrous (60.17 per cent.) than ferric, only 65.34 per cent. of ZnS was oxidized. This difference is believed, other things being equal, to have been affected by the fine grinding of the ZnS in the first case, and its absence in the other.[17]

Whatever values are given ultimately for the rate of oxidation of zinc sulphide in the presence of ferric sulphate, the results of the above experiments show beyond any doubt that the chemical equation $Fe_2(SO_4) + ZnS \rightarrow 2FeSO_4 + ZnSO_4 + S$ merely represents the chemical reaction and not the ratio. It is further shown that in contact with the oxygenated water of meteoric origin, which is no doubt an important factor in nature, even a comparatively small proportion of ferric sulphate will accelerate the rate of oxidation of the sulphide of zinc very materially. These apparent inconsistencies with the demands of the solution theory as required by the above chemical equation may be attributed to the influence of electrolytic action, as in the case with two different sulphides.[18]

The effect of ferrous sulphate upon ZnS is quite different from that of ferric sulphate. A solution of ferrous sulphate in contact with sphalerite produced no change whatsoever in either, if air is absolutely excluded. Different effects are observed, however, when the mixture is freely exposed to air. The following experiments were undertaken to demonstrate these points:

Experiment 3.—In this experiment the ferrous sulphate solution was obtained by allowing pyrites to oxidize under the influence of water and air and leaching the residue. The liquor was treated with scrap iron, and the neutral solution evaporated until a hydrate ($FeSO_4.7H_2O$) is deposited. The salt is green in color. Two portions of it, containing approximately 3.799 g. $FeSO_4$, without water, were treated in 500 c.c. of distilled water in two separate bottles (a) and (b), and each solution had a strength about $\frac{N}{10}$ of its molecular weight. The solution was then mixed with 4.873 $\frac{N}{10}$ g. of sphalerite, previously ground, and treated with 500 c.c. water.

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[16] See analyses 2, 3, 4, 5, in Table I.

[17] *Economic Geology*, vol. v, p. 35 (1910).

[18] *Economic Geology*, vol. vii, p. 28 (1912).

The mouth of the bottle (a), after mixing, was closed with its glass stopper, and the solution was left in the bottle undisturbed for four weeks. In (b) the water used was collected from rain and the mixture was allowed to stand in the bottle uncovered and freely exposed to air for the same period of time.

At the end of this period, each mixture was filtered separately. Both the residue and the filtrate in each case, were subjected to analysis.

In the analysis of (a) the filtrate was analyzed for Fe and Zn, but no Zn was found and only ferrous salts were present in solution indicating that no oxidation had occurred. The residue was still in the form of ZnS, as revealed by qualitative tests.

The analysis of (b) gave, from the residue,

	Grams		Grams		Grams
Fe_2O_3	0.7343	= Fe	0.2569	= Limonite	= 0.9824
S	1.3671	= S	1.3671	= Sulphur	= 0.1554
Zn	2.4710	= Zn	2.4710	= ZnS	= 3.6827
Impurities	0.0137	= Imp.	0.0137	= Imp.	= 0.0137
H_2O	0.2251	= H_2O	0.2481*		
	4.8112				4.8342

and from the filtrate,

Fe	1.1410	= $FeSO_4$	= 3.1030
SO_4	3.1142		
Zn	0.7846	= $ZnSO_4$	= 1.4368
	5.0398		5.0398

The Fe_2O_3 is a yellowish red precipitate in the flask. After filtering, it was added to the residue. It is believed to be a hydrated iron oxide such as limonite, since the amount of water found in the residue is only 10 per cent. less than what is theoretically required for the amount of Fe_2O_3 (0.7343) to be limonite. The results given above, show: (1) the partial oxidation of sphalerite due to free access of air; (2) that the ferrous sulphate may, under the influence of air, be oxidized to hydrated ferric oxide, and enable more sphalerite to be oxidized by giving up its sulph-oxygen ions to the latter, as evidenced by the free sulphur recalculated in the mineral component column for the residue; (3) that ferrous sulphate does not oxidize sphalerite—the analysis of (a)—but if it has free access to air, it may form hydrated ferric oxide and subsequently oxidize all the Zn into sulphate, leaving its sulphur in the free state, if the mixture has been allowed to stand long enough in the air; (4) that the oxidation of ferrous to ferric by air in contact with sphalerite is probably exhibited, as in the case of pyrites with sphalerite;[19] and (5) that, with sufficient time, all ZnS will go into solution as sulphate, and all the ferrous iron will oxidize to ferric oxide, which in the presence of water yields limonite or some other iron hydrate.

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* This figure is calculated from the formula of limonite, assuming that the Fe is in the form of that mineral.

[19] V. H. Gottschalk and H. A. Buehler. *Economic Geology*, vol. vii, pp. 22, 28 (1912).

2. *The Solubility of Sphalerite in Salt Solutions of Silver, of Copper and of Lead.*—That zinc can be replaced or carried from its mineral in form of sulphide into solution to form a salt by the replacing action of a number of metallic elements in the form of their respective salts in solution has long been known to physical chemists. In 1888, Schuermann[20] established a series in which he showed that on application of heat the zinc as sulphide is dissolved in a solution of a soluble salt of any metal higher in the series, and that zinc can be precipitated as sulphide from a solution of its salt at the expense of a sulphide of any metal lower in the series. The solutions used in his experiments were all sulphates, nitrates, or chlorides. The following series was established in the order of their desire to combine with sulphur:

1, Pt; 2, Hg; 3, Ag; 4, Cu; 5, Bi; 6, Cd; 7, Sb; 8, Sn; 9, Pb; 10, Zn; 11, Ni; 12, Co; 13, Fe; 14, As; 15, Tl; 16, Mn. The first nine metals may be considered as dissolving agents and the last six as precipitants of the zinc sulphide. The reaction may be expressed in one case, thus: $MSO_4 + ZnS \rightarrow ZnSO_4 + MS$; and in the other case thus: $MS + ZnSO_4 \rightarrow ZnS + MSO_4$; M representing any metal which either precedes or follows zinc in the series.

The metals whose minerals are commonly found in geological association with those of zinc, are iron, copper, cadmium, lead, and occasionally silver. All these metals except the first precede zinc in Schuermann's series. Electrolytically, however, all of them can be displaced by zinc, since the electromotive force of the current required to deposit each metal is less than that for the metal preceding in the electromotive series.[21]

Some results of scientific interest might be obtained by treating the sulphide of zinc electrolytically in a salt solution of any metal preceding zinc in Schuermann's series; but this is outside our present purpose. It is not without interest, however, to consider the other possible reasons for the solubility of zinc sulphide in the solution of a salt of any one metal above zinc in this series, without attributing the reaction entirely to their affinity for sulphur, as did Schuermann. With this in mind, a series of experiments were undertaken at different temperatures, degrees of concentration and time limits, in order to ascertain their relative importance, each to the other, and to observe in each case what change would take place under ordinary pressures. For this purpose two sulphates and one nitrate were taken—the three metals of which, silver, copper and lead, having, according to Schuermann, great affinity for sulphur.

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[20] Ernest Schuermann. Über die Verwandtschaft der Schwermetalle zum Schwefel. *Liebig's Annalen der Chemie*, vol. ccxlvii, p. 326 (1888).

[21] Alex. Smith. *General Inorganic Chemistry*, p. 362.

Throughout the series of experiments a mixture of definite quantity in equal or less molecular ratio of each solution was used. In each case the mixture was subject to various temperatures for a given time, and to a definite temperature for different periods of time, until the desired reaction was obtained. The solution of each metallic salt was kept at a uniform temperature before the beginning of each experiment, and the sphalerite used in all these experiments, unless indicated otherwise, was ground until it passed a 50-mesh sieve.

Experiment 4.—This comprised three tests, as follows:

(a) ZINC SULPHIDE WITH A SOLUTION OF SILVER SULPHATE

	Molecular Concentration				Temp., °C.	Time	Reaction
	ZnS		Ag_2SO_4				
	Grams	E. N.	C. C.	Norm.			
1	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	15	4 hr.	None
2	0.2436	$\frac{N}{20}$	100	$\frac{N}{25}$	25	4 hr.	Slight
3	0.1950	$\frac{N}{25}$	100	$\frac{N}{30}$	30	4 hr.	Slight
4	0.1620	$\frac{N}{30}$	100	$\frac{N}{35}$	45	5 min.	Complete
5	0.1220	$\frac{N}{40}$	100	$\frac{N}{40}$	45	5 min.	Complete
6	0.1220	$\frac{N}{40}$	100	$\frac{N}{50}$	50	4 min.	Complete
7	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	40	6 min.	Complete
8	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	35	25 min.	Apparent, but not complete
9	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	30	1 hr.	Slight
10	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	25	3 hr.	Slight
11	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	20	4 hr.	Slight
12	0.4873	$\frac{N}{10}$	100	$\frac{N}{25}$	0	20 hr.	None
13*	4.873	$\frac{N}{10}$	1,000	$\frac{N}{25}$	20	20 hr.	None
14*	4.873	$\frac{N}{10}$	1,000	$\frac{N}{25}$	30	12 hr.	Slight

E. N. = Equivalent normality. Norm. = Normality. Natural zinc sulphide and chemically pure silver sulphide were used.

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* The sphalerite in this test was in its lump form without grinding.

(b) Zinc Sulphate with a Solution of Copper Sulphate
$(CaSO_45H_2O)$

	Molecular Concentration.				Temp., °C.	Time	Reaction
	ZnS		$CuSO_4$				
	Grams	E. N.	C. C.	Norm.			
1	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	15	4 hr.	None
2	0.2436	$\frac{N}{20}$	100	$\frac{N}{20}$	25	4 hr.	None
3	0.1950	$\frac{N}{25}$	100	$\frac{N}{25}$	30	4 hr.	Slight
4	0.1620	$\frac{N}{30}$	100	$\frac{N}{30}$	45	7 min.	Almost complete
5	0.1220	$\frac{N}{40}$	100	$\frac{N}{40}$	45	7 min.	Almost complete
6	0.1220	$\frac{N}{40}$	100	$\frac{N}{40}$	50	6 min.	Complete
7	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	40	9 min.	Complete
8	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	35	25 min.	Slight
9	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	30	2 hr.	Slight
10	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	25	5 hr.	Slight
11	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	20	13 hr.	Slight
12	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	0	13 hr.	None
13*	4.873	$\frac{N}{10}$	1,000	$\frac{N}{10}$	20	13 hr.	None
14*	4.873	$\frac{N}{10}$	1,000	$\frac{N}{10}$	30	15 hr.	Slight

E. N. = Equivalent normality. Norm. = Normality. Natural zinc sulphide and chemically pure copper sulphide were used.

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* The sphalerite in this test was in its lump form without grinding.

(c) ZINC SULPHIDE WITH A SOLUTION OF LEAD NITRATE**

	Molecular Concentration				Temp., °C.	Time	Reaction
	ZnS		$Pb(NO_3)_2$				
	Grams	E. N.	C. C.	Norm.			
1	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	15	4 hr.	None
2	0.2436	$\frac{N}{20}$	100	$\frac{N}{20}$	25	4 hr.	None
3	0.1950	$\frac{N}{25}$	100	$\frac{N}{25}$	30	4 hr.	Very slight
4	0.1620	$\frac{N}{30}$	100	$\frac{N}{30}$	45	13 min.	Complete
5	0.1220	$\frac{N}{40}$	100	$\frac{N}{40}$	45	13 min.	Complete
6	0.1220	$\frac{N}{40}$	100	$\frac{N}{40}$	50	11 min.	Complete
7	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	40	18 min.	Complete
8	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	35	30 min.	Slight
9	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	30	3 hr.	Slight
10	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	25	7 hr.	Slight
11	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	20	14 hr.	Slight
12	0.4873	$\frac{N}{10}$	100	$\frac{N}{10}$	0	14 hr.	None
13*	4.8730	$\frac{N}{10}$	1,000	$\frac{N}{10}$	20	14 hr.	None
14*	4.8730	$\frac{N}{10}$	1,000	$\frac{N}{10}$	30	20 hr.	Very slight

E. N. = Equivalent normality. Norm. = Normality. Natural zinc sulphide and chemically pure lead nitrate were used.

In explanation of the results in the above three tables, it should be mentioned that the precipitated sulphides of Ag, Cu and Pb were all amorphous and that all were qualitatively determined. No attempt was made for quantitative determination.

From these results it can be readily seen that temperature alone has played a most important rôle in the solubility of ZnS at its various mol-concentrations. This is best brought out by numbers 6 to 14 inclusive in each case. It is also interesting to note from a scientific point of view, that the temperature of 45°C. seems to be a critical one in all cases. The differences in the amount of ZnS and in the concentration of the reagent do not exert any appreciable influence upon the time required to complete the reaction. The differences in time required for the solution of ZnS in each of these different solutions at different concentrations is apparently due to the difference of their metallic sulphides either in potential values or in solubility products, or both. The single potentials of the metals, Ag, Cu and Pb, in solution saturated with their sulphides, as determined by Zengelis[22] are as follows:

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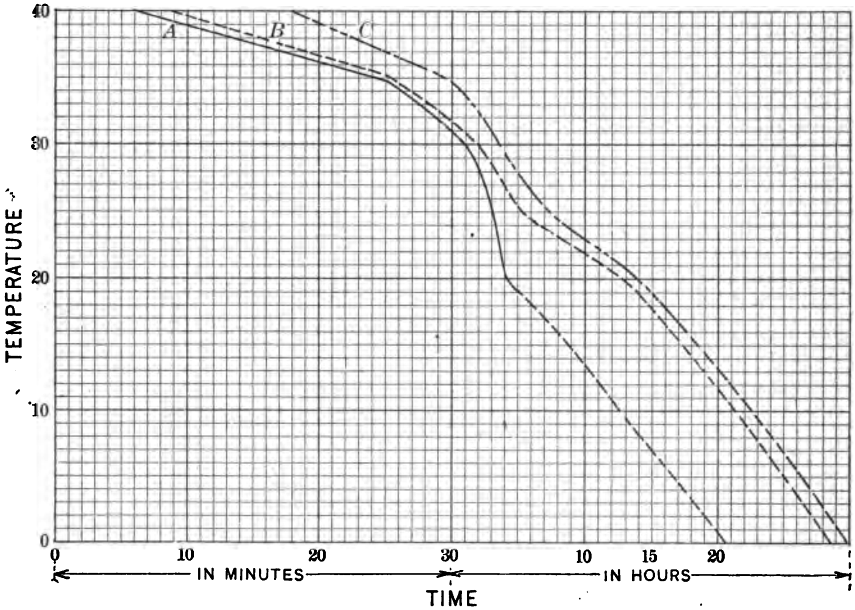
* The sphalerite in this is in its lump form without grinding.

** It was also tried with $PbSO_4$, but this compound is exceedingly difficult to dissolve, so $Pb(NO_3)_2$ was employed instead.



From 0 to 30 every small square represents one min.; from 30 to 20 to the right every small square represents one hr.

Curve *A* represents the reaction between ZnS and Ag_2SO_4.

Curve *B* represents the reaction between ZnS and $CuSO_4$.

Curve *C* represents the reaction between ZnS and $Pb(NO_3)_2$.

The broken part in each curve represents the unknown time required at a temperature between 20° and 0°C. Their concentrations are indicated in their respective tables.

Fig. 1.

Ag in Ag_2S – 0.34 volts
Cu in Cu_2S – 0.203 volts
Pb in PbS – 0.20 to 0.06 volts

The writer is unable to find any information concerning the exact potential value of zinc under similar conditions; but it is readily inferred that the potential of zinc in its zinc sulphide would be lower in comparison with that of any of these three metals from the fact that their sulphides can be precipitated from their salt solutions in contact with zinc sulphide at a definite temperature, as shown by the above experimental results. Their solubility product doubtless plays an equally important part in these reactions. The fact that less soluble compounds must be thrown down in presence of more soluble ones is too well known to be doubted.

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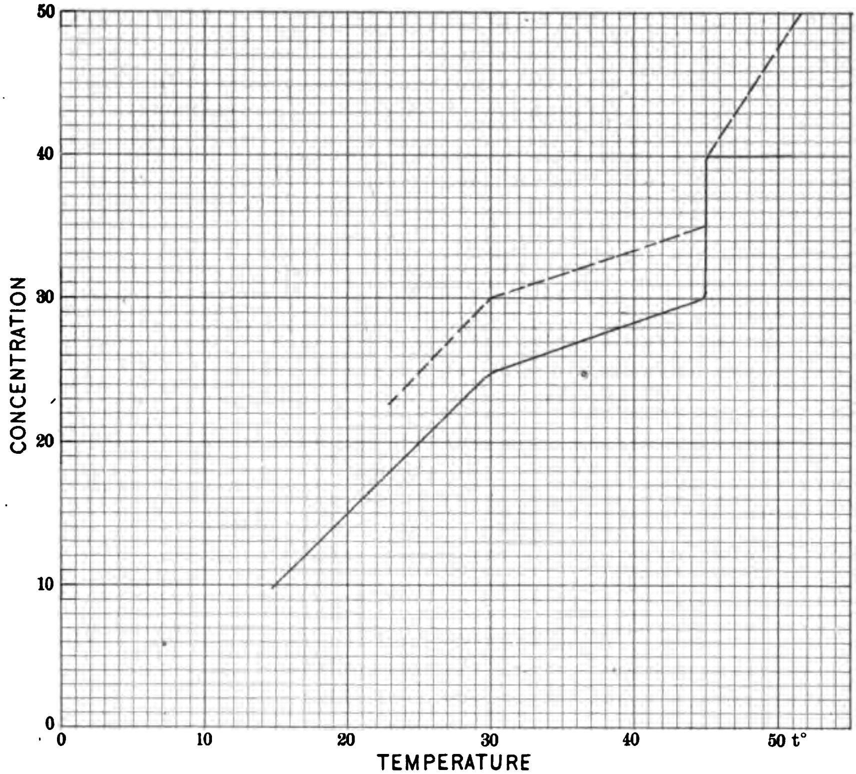
[22] K. Zengelis. *Ueber die electromotische Kräfter unlöslischer und komplexer Salze.*



The dotted line represents the behavior of Ag_2SO_4 solution with ZnS. The continuous black line represents the general behavior of $CuSO_4$ and $Pb(NO_3)_2$ solutions with ZnS.

Note the influence of these three solutions on ZnS at the temperature of 45° irrespective of their concentrations.

FIG. 2.

Fig. 1 is intended to show the time required for the reaction at various temperatures between each of these three solutions and ZnS.

Fig. 2 illustrates the general behavior of ZnS with two different salt solutions of these three metals at different concentrations and temperature, irrespective of the time required for the reaction in each case.

The above results may help to account for the fact that when zinc occurs in paying quantities as sulphide with the sulphide of Ag, Cu, or

Pb, the zinc always appears most abundantly only beneath the horizon of the minerals of the latter three, if they occur contemporaneously in the same horizon. It must not be understood that sphalerite does not occur side by side with galena or chalcopyrite or argentiferous galena. When they do occur together, it simply means there was enough sulphur to take care of all of them. This relation is particularly well shown in the occurrence of zinc-lead deposits in the Mississippi Valley, where, if galena is the chief ore, ZnS occurs only beneath the richest horizon of galena; if ZnS is the principal ore, PbS is found only in the upper portion of the orebody, as is the case with the ores in Joplin, Mo. The zinc mines at Butte[23] at the present time also show this to be true. In former times, silver sulphide occurred and was mined in the upper horizon. At present, blende is mined beneath that horizon. It is not unlikely that in these mines at Butte, copper sulphide may also occur at a greater depth in the future, but this would simply mean that the occurrences represent different periods of deposition.

With these geological observations on the one hand, and chemical relations on the other, there can be no doubt that ZnS, if present in solution together with sulphides of Pb, Cu, or Ag, is the last mineral to be formed in nature.

B. *Formation of Zinc Carbonates*

3. *The Reactions of Carbonates and Bicarbonates With Solutions of Zinc Salts.*—From the fact that the molar solubility of $ZnCO_3$ is much lower than that of $ZnSO_4$, or $ZnCl_2$, and that ZnS is often replaced in nature by $ZnCO_3$, it is readily inferred that the substances in an orebody, capable of changing ZnS to $ZnCO_3$, or precipitating $ZnCO_3$ from a solution in which Zn is held as a sulphate or chloride, are largely the soluble normal carbonates and bicarbonates.

The normal carbonates of sodium and potassium and bicarbonates of sodium and calcium were used for the following experiments to demonstrate this point.

Experiment 5.—On the precipitation of $ZnCO_3$ from its soluble compounds.

(*a*) Normal carbonate of sodium: 400 c.c. $\frac{N}{50}$ solution of $ZnSO_4$ was mixed with an excess of Na_2CO_3 solution of same normality. A flocculent precipitate formed immediately. Similarly 400 c.c. $\frac{N}{50}$ solution of $ZnCl_2$ was mixed with an excess of Na_2CO_3 solution and a precipitate likewise formed immediately. Each of these mixtures ($ZnSO_4 + Na_2CO_3$ and $ZnCl_2 + Na_2CO_3$) was then warmed and each precipitate was filtered and washed until the washings did not show any presence of Cl or SO_4.

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[23] Learned orally from Prof. J. F. Kemp, who has just visited some of the zinc mines at Butte, Mont.

The precipitate from ($ZnSO_4 + Na_2CO_3$) weighed 0.4423 g. and that from ($ZnCl_2 + Na_2CO_3$) 0.4322 g. Both were amorphous and each yielded the following after being dried in the oven and cooled in the desiccator:

The precipitate from the reaction $ZnSO_4 + Na_2CO_3$:

Chemical Components	Grams	Per Cent.	Molecular Weight	Molecular Ratio
ZnO	0.33320	75.34	81.40	0.9255
CO_2	0.06003	13.57	44.00	0.3084
H_2O	0.04912	11.21	18.00	0.6227
	0.44235	100.12		

Molecular proportions:

$0.9255 : 0.3084 : 0.6227 = 3 : 1 : 2 = 3ZnO.CO_2.2H_2O$.

The precipitate from the reaction $ZnCl_2 + Na_2CO_3$:

Chemical Components	Grams	Per Cent.	Molecular Weight	Molecular Ratio
ZnO	0.32560	75.33	81.40	0.9252
CO_2	0.05866	13.59	44.00	0.3088
H_2O	0.04799	11.10	18.00	0.6166
	0.43225	100.02		

Molecular proportions:

$0.9252 : 0.3088 : 0.6166 = 3 : 1 : 2 = 3ZnO.CO_2.2H_2O$.

The above results clearly show that each of these precipitates represents, not a normal zinc carbonate, but a basic zinc carbonate such as hydrozincite. Its reaction may be represented by some such equation as this:

$$3Na_2CO_3 + 4H_2O + 3ZnSO_4 \rightarrow 3Na_2SO_4 + 2H_2CO_3 + ZnCO_3.2Zn(OH)_2$$
$$3Na_2CO_3 + 4H_2O + 3ZnCl_2 \rightarrow 6NaCl + 2H_2CO_3 + ZnCO_3.2Zn(OH)_2$$

Moreover, Dana, in his *System of Mineralogy,* suggested this formula for hydrozincite. According to him, the constituents of a hydrous zinc carbonate from Auronz, were as follows: ZnO, 73.21; CO_2, 14.55; and H_2O, 11.38 per cent.

In another experiment, the reactions between Na_2CO_3 and $ZnSO_4$ solutions gave likewise a precipitate of basic zinc carbonate, but it was found to contain only one molecule each of $ZnCO_3$ and $Zn(OH)_2$. This difference probably indicates that the formation of basic zinc carbonate varies in composition according to conditions.[24]

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[24] Alex. Smith. *General Inorganic Chemistry,* p. 649.

The precipitate obtained from the reaction of potassium carbonate with $ZnSO_4$ or $ZnCl_2$ was identical in appearance, but no analysis was made.

Duplicates of this experiment between $ZnSO_4$ and Na_2CO_3 were performed at the same time, and under same conditions; but instead of filtering off the precipitate, carbon dioxide was allowed to pass directly into the mixture for half an hour to one hour. The mixture (formerly precipitate and liquid) became in this time a clear solution showing that $ZnCO_3$ is not stable in presence of free H_2CO_3. This agrees with the fact that in the waters of zinc mines, where zinc carbonates occur, no H_2CO_3 has been reported.

(*b*) 1. Bicarbonate of Sodium ($NaHCO_3$): To an excess of this bicarbonate solution was added 400 c.c. of $\frac{N}{50}$ $ZnSO_4$ solution. The mixture soon became milky and a very fine precipitate was formed. Some CO_2 gas appeared during the precipitation. In the same way $ZnCl_2$ was treated with $NaHCO_3$ and the same kind of precipitate was formed. Each mixture was heated almost to boiling in order to drive off the excess of CO_2. Each precipitate was separated from its liquid by filtration, and was washed until no sign of SO_4 or Cl was shown. The precipitate in each case was amorphous.

The weight of the precipitate from $ZnSO_4$ was 0.5112 g., and that from $ZnCl_2$, 0.4873 g. After being dried in the oven and cooled in the desiccator they yielded the results given below:

	The precipitate from the reaction between $ZnSO_4$ and $NaHCO_3$		The precipitate from the reaction between $ZnCl_2$ and $NaHCO_3$	
	Grams,	Per Cent.	Grams,	Per Cent.
ZnO	0.3302 =	64.6	0.3164 =	64.94
CO_2	0.1798 =	35.18	0.1712 =	35.14
H_2O	trace		none	
Total	0.5100	99.78	Total..... 0.4876	100.08

The substance is evidently anhydrous carbonate of zinc and this reaction may be represented by the following simple equation:

$$\begin{array}{l} ZnSO_4 + 2NaHCO_3 \rightarrow ZnCO_3 + Na_2SO_4 + \underset{\displaystyle H_2O + CO_2}{\underset{\downarrow}{H_2CO_3}} \\ ZnCl_2 + 2NaHCO_3 \rightarrow ZnCO_3 + 2NaCl + \underset{\displaystyle H_2O + CO_2}{\underset{\downarrow}{H_2CO_3}} \end{array}$$

Theoretically anhydrous carbonate of zinc contains: ZnO, 64.8; CO_2, 35.2 per cent.

(*b*) 2. Bicarbonate of Calcium: 1.001 g. of finely powdered pure limestone (an equivalent of $\frac{N}{50}$) were placed in 1,000 c.c. water, into which carbon dioxide was subsequently allowed to pass until the powdered mineral entirely disappeared, being converted into the more soluble bicarbonate. The reaction may be represented by the following equation:

$$CaCO_3 + H_2CO_3 \rightarrow Ca(HCO_3)_2$$

400 c.c. of this lime solution were mixed with a slight excess of $\frac{N}{50}$ solution of $ZnSO_4$ in one flask and $ZnCl_2$ in the other. Each mixture immediately became milky and a very fine precipitate formed. The liquid was almost neutral. After being heated to boiling each precipitate was filtered off, washed until the last few drops of washing water no longer showed the presence of SO_4 or CO_3 ions. Then each was dried in the steam oven. They were weighed, after they had been allowed to cool in the desiccator, and yielded respectively:

A. The precipitate from Ca $(HCO_3)_2$ + $ZnSO_4$ weighed 0.4581 g.

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
CaO	0.11290	24.63	56.1	0.4389	1
CO_2	0.17862	38.98	44.0	0.8700	2
ZnO	0.10650	36.34	81.4	0.4287	1

B. The precipitate from $Ca(HCO_3)_2$ + $ZnCl_2$ weighed 0.4272 g.

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
CaO	0.11070	25.90	56.1	0.4616	1
CO_2	0.10743	39.19	44.0	0.8906	2
ZnO	0.14910	34.90	81.4	0.4287	1

Their mineral components according to the amount of the above chemical analyses, may be represented respectively:

(A)	(B)
$CaO.CO_2$ or $CaCO_3$ = 0.2014 g.	$CaCO.CO_2$ or $CaCO_3$ = 0.1970 g.
$ZnO.CO_2$ or $ZnCO_3$ = 0.2566 g.	$ZnO.CO_2$ or $ZnCO_3$ = 0.2320 g.

From this it appears that the compound consists of carbonates of calcium and zinc. From the amount of calcium carbonate used at the start, before its conversion of $Ca(HCO_3)_2$, and from the amount of the mineral component of the same compound, it is readily seen that the amount of Ca present in 400 c.c. of $\frac{N}{50}$ $Ca(HCO_3)_2$ solution is 0.1614 g. and that the metal recovered in its original form of $CaCO_3$ by interaction with $ZnSO_4$ is 0.0808 g. or a little over half of its original amount.

It is also theoretically known that one molecule of $Ca(HCO_3)_2$ in solution is able to precipitate an equivalent of $ZnCO_3$, and to produce a complete reformation of $CaCO_3$, provided they are not affected by the byproduct H_2SO_4, as indicated in the equation:

$$(1)\quad Ca(HCO_3)_2 + ZnSO_4 \rightarrow CaCO_3 + ZnCO_3 + H_2SO_4$$

But this hardly appears to be correct, as shown by the amount of the mineral components which is only about half of what this equation demands. Suffice it to say that the formation of the "secondary" H_2SO_4 (or HCl, if $ZnCl_2$ is used instead of $ZnSO_4$) must be used up in its reaction with the chemically precipitated $ZnCO_3$, and $CaCO_3$, thus converting part of both of these precipitates into $ZnSO_4$ and $CaSO_4$ on the one hand, and giving rise to the formation of H_2CO_3 on the other until the new equilibrium is reached. These relations may be best shown by the following equation:

$$(2)\quad H_2SO_4 + \tfrac{1}{2}CaCO_3 + \tfrac{1}{2}ZnCO_3 \rightarrow \tfrac{1}{2}CaSO_4 + \tfrac{1}{2}ZnSO_4 + H_2CO_3$$

As soon as this condition is reached the rest of the $CaCO_3$ and $ZnCO_3$ will be able to remain as precipitates, since these carbonates are quite stable in the solution of their sulphates. It is also possible that the carbonates of Zn and Ca may co-exist as one compound, analogous to dolomite, whose formula is ordinarily written: $MgCO_3.CaCO_3$.

The H_2CO_3 shown in equation (2) doubtless breaks up into H_2O and CO_2. These relations are further proved by the qualitative test of the filtrate which, after being heated to boiling for 15 min., showed only the presence of Ca, Zn and SO_4 ions.

These reactions may help to account for the occurrence of zinc carbonate in, and in contact with, limestone in nature, as in the Bertha Mines of Virginia.

Dr. Watson[25] has suggested that the calcium carbonate must be first converted into soluble calcium bicarbonate due to "the interaction of the percolating atmospheric waters containing carbon dioxide in solution on limestone," before the reaction to form zinc carbonate takes place. This is in full accord with the writer's experience in the laboratory. He further suggested that in the formation of normal zinc carbonate, under natural conditions, the compound is probably in the form of basic carbonate as an "intermediate step" towards the formation of normal zinc carbonate from the reactions between zinc sulphate and calcium bicarbonate. But the writer's laboratory experience tends to indicate that normal zinc carbonate only is formed if the reagent is a bicarbonate, whereas a basic carbonate of zinc only is precipitated in varying composition if the reagent is a normal carbonate.

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[25] T. L. Watson. *Bulletin No.* 1, *Geological Survey of Virginia,* p. 44 (1905).

The precipitates from duplicates were not filtered, but directly treated with carbon dioxide, which was allowed to pass into the mixture for about three-quarters of an hour. At the end of this period the precipitates had entirely disappeared and the liquid had become clear, indicating that in presence of free H_2CO_3 the normal carbonate of zinc cannot exist, probably just as the calcium carbonate became soluble, forming bicarbonate, under the same condition, as shown in the preparation of it for the above experiments.

Similar experiments were performed with chemically made sulphide of zinc and with natural sulphide from Joplin, Mo. In these experiments 0.4873 g. (the equivalent of $\frac{N}{100}$) of each of these sulphides was treated in 1,000 c.c. water and mixed with a double amount of $\frac{N}{100}$ solution of normal sodium carbonate in one flask, and of bicarbonates of sodium and calcium in the other, as before. No evidence of appreciable change in each case was noted in the course of two months. Carbon dioxide was then passed into each of these mixtures for an hour, but evidence of change was equally absent in each case. This suggests that direct oxidation of sphalerite to zinc carbonate is far from being realized under laboratory conditions.

Experiment 6*a*.—On the Conversion of Zinc Carbonate from Zinc Sulphide. These experiments were repeated, but only the natural sulphide of zinc was used, with modifications in the conduct of the experiments. 100 g. of coarsely crushed (not powdered) sphalerite was spread in layers in a large glass jar (*A*, Fig. 3), 9 in. high and 4 in. in diameter, with alternate layers of various sizes of sandstones, shales and limestones above and below the sulphide layers. The entire charge filled the jar three-quarters full. A solution of $\frac{N}{50}$ $CuSO_4 \cdot 5H_2O$ was first allowed to flow slowly into this jar through a glass tube (T_1) and as soon as the solution filled the jar up to three-quarters of the depth, the solution was allowed to flow through another glass tube (T_2), which was fixed to reach downward one-fourth of the depth of the jar, into another bottle which served as a receiver (*C*) containing the excess of the solution of $CuSO_4$, which was returned to the supplying bottle (*B*) from time to time. Altogether 18,000 c.c. of $\frac{N}{50}$ $CuSO_4$ solutions were used during the entire period, including fresh portions of this solution, added from time to time to compensate for evaporation. This treatment was continued for a period of four months, under ordinary conditions of temperature and pressure.

At the end of this period the colorless solution and the blackish precipitate, formed during the operation in the jar (*A*), and the solutions in bottles (*B*) and (*C*), amounting by this time to about 3,400 c.c., were all emptied into a large flask. The sandstone, shale and limestone together with the remaining lumps of sphalerite in jar (*A*), were carefully washed with water until the washing water was free from SO_4 ions. All the washings were added to the mixture.

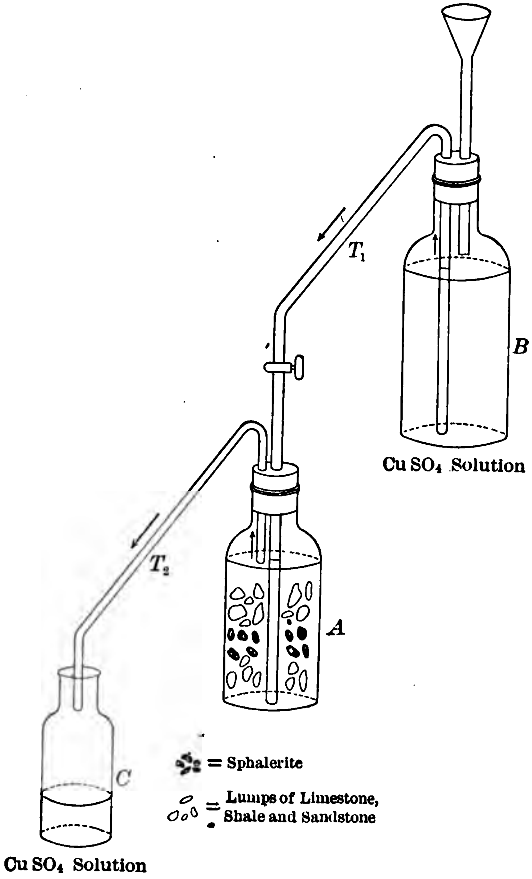

Fig. 3.—Apparatus for Experiment 6*a*.

The blackish precipitate was then filtered off from this mixture and washed until the washings showed no presence of SO_4 ions. These washings were also added to the filtrate which was then made up to 5,000 c.c. with distilled water.

It was the purpose of this experiment to ascertain in a quantitative way: (1) whether copper as sulphate in dilute solutions would attack the natural ZnS in contact with limestone, shale and sandstone; and (2) whether such reagent would also dissolve part of those rocks in the jar, apart from sphalerite, in the course of circulation. To determine these two points, definite portions of the filtrate and the precipitate, whose weight after being dried in the desiccator was found to be 17.2 g., and which contained some free sulphur, were taken for analysis. The filtrate, according to litmus paper tests, was slightly acidic or almost neutral; 500 c.c. of it were used in the analysis recorded below: The precipitate yielded:

				Amount in Precipitate and in Filtrate	Amount of Cu and SO_4 Originally Present in Solution
ZnO	2.1790	Zn	1.7520	1.7520	....
Cu	11.4400	Cu	11.4400	11.4400	11.4400
CO_2	1.1790	....	....	....	....
S including free S	5.7600	gangue	7.4662	7.4662	....
SiO_2	0.1002	....	....	....	....
	20.6582		20.6582		
and from the filtrate were obtained					
SiO_2	0.0018	SiO_2	0.0018	....	....
Al_2O_3	0.00366	Al	0.00194	....	....
FeO	....	....	....	....	....
Fe_2O_3	0.00844	Fe	0.00590	....	....
CaO	0.03673	Ca	0.02625	....	....
MgO	0.06253	Mg	0.03775	....	....
Na_2O	0.00824	Na	0.00612	....	....
K_2O	0.01250	K	0.01038	....	....
SO_4	1.72810	SO_4	1.72820	17.2820	17.2820
ZnO	1.24700	Zn	1.00200	10.0200	....
				47.9602	28.722

From the molecular weight, it is known that 18,000 c.c. $\frac{N}{50}$ $CuSO_4.$-$5H_2O$ solution furnish about 11.448 g. Cu, and from the chemical reactions,

$$CuSO_4 + ZnS \rightarrow CuS + ZnSO_4$$

the amount of copper (11.440 g.) actually found in the precipitate would require, by calculation, about 5.76 g. sulphur to form CuS.[26] The amount of zinc (1.7520 g.) was undoubtedly in the form of carbonate, precipitated from zinc sulphate, as may be inferred also from the presence and amount of carbon dioxide in the analysis. Thus by calculation, according to the above analysis, the precipitate contains the following:

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[26] The CuS is believed to have been reduced to Cu_2S and the free S in the precipitate is due to this reduction.

	Grams		Grams
CuS =	17.20	= Cu	11.4400
$ZnCO_3$ =	3.358	= Zn	1.7520
SiO_2 =	0.1002	= gangue	7.4662
	20.6582		20.6582

The analysis of the filtrate, through the absence of Cu, points to its precipitation from solution, and the presence of an almost equivalent amount, molecularly speaking, of zinc in the solution as sulphate, indicates the oxidation of zinc from its sulphide. Lastly, the presence of other substances, which were absent in the mixture at the beginning of the experiment, indicates that a solution containing sulphate of copper (or a soluble salt of any other metal beneath hydrogen in the electromotive-force series) tends to carry many common rock-making minerals or components into solution and, as a consequence, neutralize or decrease its acidic character to a very appreciable extent.

It is also known from the molecular weight that 18,000 c.c. $\frac{N}{50}$ solution of $CuSO_4$ furnish 11.448 g. Cu and 17.2818 g. SO_4.[27] Theoretically, a $\frac{N}{50}$ solution of cupric sulphate in one liter requires 0.654 g. Zn to displace all the Cu.

But from the fact that Cu is entirely absent in the filtrate, as shown by the above analysis, the theoretical amount of Zn oxidized into sulphate or required to replace the amount of Cu actually found in the precipitate, cannot be less than 11.76 g., as based on the reaction,

$$CuSO_4 + ZnS \rightarrow CuS + ZnSO_4$$

By deducting 11.76 g. (to satisfy the replacement of the Cu) from the amount of Zn found in the filtrate (10.02 g.) and in the precipitate (1.752 g.) there is an excess of 0.012 g. (11.772 − 11.76 = 0.012), which is about 0.1019 per cent. more than what is theoretically necessary for the amount of copper as found in the analysis of the precipitate. The difference in these values (between theoretical and practical calculations) is undoubtedly due to errors in manipulation.

The amount of SO_4 in the original solution as represented by $CuSO_4$ at start was 17.2818 g. (theoretically).[27] The amount recovered was

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[27] In calculating the SO_4 from the sulphate of copper no water of crystallization was considered.

17.282 g. After deducting from the total amount of SO_4 (17.282 g.) the theoretical amount required for all the Zn as sulphate found in the filtrate, there remains 2.552 g. (17.282 − 14.73 = 2.552) which may be safely assigned to the other bi- and tri-metals found in the filtrate.

That the sulphide of zinc is oxidized by the solution of copper sulphate is practically certain. The remaining solution and the washings, chiefly containing Zn as sulphate, are now ready for the treatment as used in experiment 6*b*.

Experiment 6*b*.—The precipitation of $ZnCO_3$ from the oxidized solution of ZnS: 500 c.c. of this mixed solution was treated with an excess of $\frac{N}{50}$ solution of $NaHCO_3$. A very fine precipitate was formed immediately. The whole mixture was gently heated almost to boiling; some CO_2 was noted passing off during the heating. The mixture was allowed to cool and the precipitate was separated from the liquid by filtration. The former was washed until the washings showed no sign of the presence of SO_4. It was then dried in the steam oven and cooled in the desiccator before the precipitate was weighed (1.9884 g.) and analyzed. The following are the analytical results:

	Gram	Per Cent.
SiO_2	0.0019	0.095
CaO	0.0367 (=Ca, 0.02625)	1.842
MgO	trace	....
H_2O	none	....
CO_2	0.7105	35.260
ZnO	1.2470 (=Zn, 1.002)	62.560
	1.98838	99.757

From the above analysis, it may be safely assumed that all the chemical components present (except SiO_2) were in the form of carbonates, since, by recasting for their carbonates it is found, as shown below, that the amount of CO_2 required by these by-metals (Zn and Ca) is 0.7028 g. Namely,

0.02625 g. Ca will form 0.0655 g. $CaCO_3$ with 0.0288 g. CO_2
1.0020 g. Zn will form 1.9210 g. $ZnCO_3$ with 0.6740 g. CO_2

0.7028 g.

The remaining CO_2 (0.0077 g.) is probably in combination with MgO; but the latter is too small in amount to be considered. The amount of these mineral components as carbonates is obtained by calculation from their chemical components. Their mineral component proportion may be represented as follows:

	Grams	Per Cent.
$CaCO_3$	0.0655	3.294
$MgCO_3$	trace	.....
$ZnCO_3$	1.9210	96.61
		99.904

Since there is enough CO_2 to take care of all the zinc found, and no water is present, and SiO_2 is too small in amount to be considered as having combined with Zn in any way, the Zn compound is, beyond doubt, a normal carbonate.

In another experiment a mixture of dilute solutions of bicarbonate was allowed to pass through a jar containing sphalerite in the manner used for the experiments under 6*a*. The operation was continued for two months, but met with no success.

In the light of the above results, it seems quite certain that $ZnCO_3$ is not formed direct from its sulphide. The sulphide is first oxidized into zinc sulphate, and the latter may migrate a short or long distance, as the case may be, before the carbonation takes place. In the waters which percolate through strata in which ZnS (sphalerite) occurs, the chemically active compounds must be complex, and must be provided with oxidizing material of one kind or another. These waters will doubtless oxidize the sphalerite at an incipient stage to sulphate. The oxidized part will, according to existing conditions, probably be transported to some other place until it meets an alkaline solution or solutions of alkaline earths, which in nature mostly contains soluble carbonates. The deposition of $ZnCO_3$ will then take place, if the metals like Pb or Cu, which will form more insoluble carbonates than Zn, are either absent or very small in amount. These conditions will account for the fact that both $ZnCO_3$ and ZnS have been found in different horizons as if they had no relation to each other, as in the Circle Mines at Oronogo, Mo.[28] and in some zinc mines in Virginia.[29]

In case an orebody of sphalerite is attacked by a neutral solution of sulphates and carbonates, the sphalerite may be first oxidized into sulphate, especially when the metals above zinc in Schuermann's series[30] are present as sulphates in the solution, and then immediately carbonated, without suffering the removal of any appreciable quantity by migration. This may help to account for the fact that sphalerite is sometimes closely associated with smithsonite, and the latter always grades downward into the former.

4. *The Reaction of Zinc Sulphate Solution with Limestones.*—The

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[28] The writer had an opportunity in the summer of 1913 to visit some of the zinc mines in southwest Missouri, particularly in the Joplin district. All the writer's own field data were obtained at that time.

[29] T. L. Watson. *Bulletin No.* 1, *Geological Survey of Virginia*, p. 45 (1905).

[30] *Liebig's Annalen der Chemie*, vol. ccxlvii, p. 326 (1888).

Experiment 7: *With Crystalline Limestone**

	Analysis of the Limestone				Analysis of the Solution	
	Before Treatment		After Treatment†		Before Treatment	After Treatment
	Grams	Per Cent.	Grams	Per Cent.	Grams	Grams
SiO_2	0.0427	0.6521			CaO	0.6063 = 1.472 $CaSO_4$
Al_2O_3	0.0234	0.3574			SO_4 2.597	2.5967
Fe_2O_3	0.0183	0.2796			ZnO or	1.3200 = 2.619 $ZnSO_4$ or
CaO	3.543	54.1200 = 96.54 $CaCO_3$	2.9380	43.1200 = 76.93 $CaCO_3$	Zn 1.768	1.0470
MgO	0.0193	0.2947 = 0.6155 $MgCO_3$	0.0156	0.2290 = 0.4769 $MgCO_3$		
CO_2	2.800	42.7408	2.7966	41.0400		
ZnO			0.8800 or	12.9100 = 19.9000 $ZnCO_3$		
Zn			0.7071			
	6.4467	97.1655		97.3069		

Experiment 8: *With Impure Limestone*

	Analysis of the Limestone				Analysis of the Solution	
	Before Treatment		After Treatment		Before Treatment	After Treatment
	Grams	Per Cent.	Grams	Per Cent.	Grams	Grams
Insoluble and organic matter	3.432				CaO	1.16
SiO_2	2.406				MgO	0.2813
Al_2O_3	3.512				SO_4 2.597	2.597
Fe_2O_3	0.9223				ZnO 1.768	
CaO	13.5200	34.26 = 61.12 $CaCO_3$	12.380	31.37 = 55.94 $CaCO_3$		
MgO	1.9680	4.987 = 10.42 $MgCO_3$	1.687	4.275 = 8.933 $MgCO_3$		
H_2O+	0.3117					
H_2O-	0.1947					
CO_2	12.751		12.735			
MnO	trace					
Na_2O	0.1270					
K_2O	0.325					
ZnO			2.201	7.158 = 11.03 $ZnCO_3$		
Zn			1.750			
	34.4697	71.54				

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* This limestone was obtained by Dr. C. P. Berkey from Rockland. Dr. Berkey gave $CaCO_3$, 96.45, and $MgCO_3$, 0.12 per cent. in one case; and $CaCO_3$, 94.9 and $MgCO_3$, 20 per cent. in the other.

† H_2O and organic matter was not determined. Only carbonates determined.

solution of zinc sulphate used in the following experiments is all $\frac{N}{20}$, that is, 4.365 g. $ZnSO_4$ or 1.768 g. metallic zinc in every 1,000 c.c. The limestones used were analyzed before and after the treatment, which in each case was kept going for a period of four months in a closed bottle, to which air was occasionally admitted through a side tube. In each case a definite weight of limestone was treated with 1,000 c.c. $\frac{N}{20}$ solution of $ZnSO_4$. At the end of four months the limestone in each case was found to have been covered with a white thin coat which, upon drying, appeared to be somewhat powdery, and which was $ZnCO_3$ as shown in the respective analyses recorded on preceding page.

From the above analyses, it is shown that in the case of crystalline limestone, only two-thirds of the zinc was precipitated as $ZnCO_3$, while with the impure limestone, zinc is entirely absent in the sulphate solution after the treatment. These apparent differences from the treatment of two different limestones with the same solution of $ZnSO_4$ are believed to be due to the presence of organic matter, which is common in impure limestone.

It is further shown that carbonates of both magnesium and calcium can be replaced by zinc under the conditions described above.

In another experiment with the impure limestone, minute particles of what appeared to be ZnS could be seen with the aid of a powerful lens. Such precipitation of the zinc as sulphide, by commingling of the oxidizing and reducing solutions in limestones, is not only possible, but also probable, according to some geologists, notably Dr. Buckley,[31] in his studies of the origin of zinc sulphide in the Joplin district. The evidence that these minute particles were ZnS, was, however, not conclusive, for the reason that it was difficult to separate them from the whitish coating of zinc carbonate. An attempt was made to prepare thin sections, but the replaced material was too soft for grinding.

C. *Peculiar Conditions Affecting the Formation of Zinc Silicate (Calamine) and Zinc Carbonate*

5a. The Effect of Carbon Dioxide upon the Basic Silicate and the Carbonates of Zinc in Water, and its Bearing upon the Conditions of their Precipitation.—Hydrous silicate of zinc[32] has long been known to be soluble in water in presence of CO_2 gas, but no one has attempted to explain in the light of modern chemistry the bearing of the latter upon the formation of this silicate and the association of the silicate with carbonate of the same metal. The following experiments were undertaken in this connection.

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[31] E. R. Buckley. *Types of Ore Deposits*, pp. 103 to 132.

[32] In 1849, Victor Monheim made his quantitative determination of zinc silicate by dissolving it in water in presence of carbon dioxide. *Verhandlungen des Naturhistorischen Vereius*, vol. vi, pp. 1 to 24.

Experiment 9*a*.—5 g. of pure calamine from the vicinity of Chihuahua, Mexico, secured from the collection of ores in the Department of Geology at Columbia University, were powdered until they passed 100-mesh sieve. This amount was mixed with 1 liter of water in a 2-liter flask. A stream of carbon dioxide was allowed to pass through for 80 hr., with occasional shaking of the contents. The mixture was filtered and a residue collected and washed before it was dried in the oven. The washings were added to the filtrate, which was perfectly clear. After being dried in the oven and cooled in the desiccator, the residue was weighed at 3.48 g. For convenience it is designated as first residue.

The filtrate was heated to boiling for about half an hour in order to drive off the CO_2 completely. A fine precipitate came down, which was also filtered, washed and dried in the oven. Its weight was found to be 1.462 g. It is designated as second residue. There is a loss of 0.012 per cent. due to manipulation.

By analysis these residues yielded respectively:

First Residue	Grams	Per Cent.	Second Residue	
			Gram	Per Cent.
ZnO	2.3480	67.48	0.9868	67.50
SiO_2	0.8714	25.03	0.3661	25.03
H_2O	0.2597	7.463	0.1091	7.461
		99.973		99.991

That the hydrated silicate of zinc is soluble in water in presence of enough carbon dioxide, and that the same silicate can be reconverted into its solid form from such solution by raising the temperature, there is no doubt. It is also believed that the carbon dioxide does not combine with zinc to form some such soluble compound as bicarbonate of zinc, for the H_2O in the zinc silicate is basic,[33] and if bicarbonate were formed, zinc carbonate would come down as precipitate when the CO_2 was driven off by heating. Thus the carbon dioxide merely exerts its influence as a catalytic agent in carrying the silicate of zinc into solution, without affecting any change in its molecular combination.

Experiment 9*b*.—The next question is whether or not an impure calamine, that is a mixture of zinc silicate and zinc carbonate with carbonates of some other bi-metals will also become soluble in water in excess of carbon dioxide as shown in each individual case. For this purpose a piece of impure calamine, obtained from a miner in Joplin, Mo., was chosen for the experiment.

In order to know the exact contents, 3.259 g. of the powdered material from the specimen were analyzed, with the following results:

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[33] Dana. *System of Mineralogy*, Sixth Edition, p. 548.

Chemical Components			Mineral Components		
	Grams	Per Cent.		Grams	Per Cent.
SiO_2........	0.5622	17.300	Calamine	2.2470	69.130
Fe_2O_3.......	0.0377	1.160	Zinc carbonate	0.7632	23.480
FeO........	trace	....	Calcium carbonate	0.1378	
CaO........	0.0772	2.375	Magnesium		4.240
			carbonate.......	0.0786	2.418
MgO........	0.0376	1.157	Limonite	0.0443	1.363
CO_2........	0.3673	11.310	....	....	....
ZnO........	2.0110	61.870	....	....	....
H_2O........	0.1758	5.409	....	....	....
Na&K......	trace	....	....	....	....
MnO.......	trace	....	....	....	....
P_2O_5........	none	....	....	....	....
	3.2688	100.581		3.2709	100.631

An equal amount (3.259 g.) of this material was taken for the treatment in the experiment, which was in every respect the same as in the last case. At the end of the experiment, the mixture was filtered. Upon boiling the filtrate, as before, a heavy precipitate collected. After being dried in the oven and cooled in the desiccator, it was found to weigh 2.522 g. and yielded:

Chemical Components				Mineral Components			
	Grams	Per Cent.	Per Cent.		Grams	Per Cent.	Per Cent.
SiO_2......	0.4109	16.2900	12.6400	Calamine......	1.6412	65.0700	50.4900
Fe_2O_3.....	0.0107	0.4243	1.2650	Zinc carbonate.	0.7311	28.9900	22.5000
FeO							
CaO......	0.0411	1.6290	1.0490	$CaCO_3$.........	0.0734	2.9110	2.2500
MgO.....	0.0341	1.3520	0.3293	$MgCO_3$.......	0.0713	2.8280	2.1940
CO_2......	0.3261	13.2000	10.0300	Limonite......	0.0126	0.4997	1.3377
ZnO......	1.5825	63.2800	48.7000				
H_2O......	0.1243	4.9280	3.8230				
	2.5297	99.1033	77.8363		2.5296	100.2987	77.8297

The chemical components were calculated, by recasting, into mineral components for the purpose of finding the exact amount of each mineral component dissolved in water in presence of the carbon dioxide. Its percentage was then recalculated (column 3) on the basis of the original weight of the material used in the experiment.

By comparing the figures in column 3 with the mineral percentage in the analysis of the specimen it can be readily seen: (1) that in the presence of carbon dioxide, nearly all the carbonates in the specimen were carried into solution; (2) that the zinc carbonate was almost entirely dissolved; and (3) that a much greater percentage of zinc silicate was dissolved from the impure specimen than from the pure one used in experiment 7*a*. It is, therefore, plausible that the presence of the impurities in the specimen, particularly $ZnCO_3$, must have exerted some influence upon the solution of the zinc silicate. It is at least suggestive that the molecules of $ZnCO_3$, which were present with those of the hydrous silicate of zinc in the material used, would attract molecule by molecule the components of the silicate in water under the influence of carbon dioxide, but would not necessarily effect any change in their respective molecules or molar-volumes, since their equilibrium remains the same in the aqueous state under the above conditions.

5*b*. *The Reactions of Solutions of Soluble Silicate with Calamine and Smithsonite in Solution in Presence of Carbon Dioxide.*—Since the excess of carbon dioxide can hold both silicate and carbonate of zinc in solution, the question arises as to what peculiar effect will be produced when a silicate solution is introduced into such a mixture. That is to say, whether the basic silicate can be brought about by means of a soluble silicate solution from a solution of zinc carbonate and zinc silicate made by dissolving the latter two under the influence of carbon dioxide.

It has long been known that silicon dioxide may replace carbon dioxide when heated with an alkaline carbonate,[34] and inasmuch as this property of silicon dioxide may have direct bearing on the origin of zinc silicate in association with zinc carbonate such as frequently found in zinc mines, the study of its precipitating capacity was undertaken.

Experiment 10.—4 g. of finely powdered calamine obtained from Chihuahua, Mexico, and 4g. of chemically pure zinc carbonate were mixed together. The mixture (8 g. altogether) was then digested in 1,000 c.c. water under the influence of carbon dioxide, which was allowed to pass slowly (one bubble for every three seconds) through the mixture for a period of 50 hr. At the end of this period, the mixture was filtered and a residue, which for clearness, will be designated as residue X, was removed. It was dried in the steam oven and weighed 3.9692 g. By analysis the residue X, yielded:

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
SiO_2..............	0.5380	13.550	60.4	0.2244	
CO_2..............	0.6387	16.090	44.0	0.3656	3
H_2O..............	0.1603	4.033	18.0	0.2241	
ZnO..............	0.6310	66.300	81.4	0.8145	3
	3.9680	99.973			

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[34] Alex. Smith. *General Inorganic Chemistry*, p. 519.

The filtrate, after the removal of the residue X, was treated with an excess of dilute solution of Na_2SiO_3. A gelatinous precipitate, designated as precipitate Y, immediately collected and was filtered off, dried in the steam oven, and weighed, after being cooled in the desiccator, 2.3763 g.

As shown by the analytical results recorded below, it is certain that the gelatinous precipitate is not calamine ($2ZnO.SiO_2.H_2O$), but is a different substance, which according to the molecular proportions of the chemical components, as shown below, may be called zinc meta-silicate ($ZnSiO_3$ or $ZnO.SiO_2$). The following is the analysis of the precipitate Y:

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
SiO_2	1.0120	42.600	60.4	0.705300	1
H_2O	0.0003	0.012	18.0	0.000667	0
ZnO*	1.3610	57.280	81.4	0.703800	1
	2.3733	99.892			

$7038 : 0.7053 := 1 : 1 = ZnO : SiO_2 = ZnSiO_3$.

The filtrate from the precipitate Y was colloidal. It was heated to boiling for half an hour until CO_2 was no longer given off. During the heating, a precipitate collected as expected. After being cooled, the precipitate was removed by filtration from the colloidal solution and was washed with distilled water. It was dried in the oven and cooled in the desiccator, and weighed 1.9747 g. For clearness, it will be designated as precipitate Z, from which were obtained:

	Grams	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion
SiO_2	0.4634	23.470	60.4	0.3886	1
H_2O	0.1381	6.993	18.0	0.3885	1
ZnO	1.2490	63.240	81.4	0.7769	2
A gelatinous residue	0.1235	6.253			
	1.9740	99.956			

$0.7769 : 0.3886 : 0.3885 = 2 : 1 : 1 = 2ZnO.SiO_2.H_2O$.

From the molecular proportions of X, it can be readily seen that the residue consists of silicate and carbonate of zinc, as the undissolved portions of their original amount, or the proportion of the oxide of zinc is almost equal to the sum of those of the other three components, namely

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* Zn in this case was determined as pyrophosphate and ZnO was calculated from the molecular formula of $Zn_2P_2O_7$ and found to contain 1.082 grams metallic zinc.

0.2244 + 3.656 + 0.2241 = 8145 − 4 (to be exact) or the amount of SiO_2, CO_2, and H_2O = the amount of ZnO.

Thus $3ZnO = SiO_2 + CO_2 + H_2O$ or

$$3ZnO + SiO_2 + CO_2 + H_2O = 2ZnO.SiO_2.H_2O + ZnO.CO_2 \text{ (or } ZnCO_3)$$

On the basis of the original amounts used at the beginning of the experiment and by means of their molecular proportions, it is found that the residue X constitutes 2.151 g. or 53.966 per cent. of hydrous silicate of zinc and 1.82 g. or 45.5 per cent. of the normal carbonate of zinc as representing the undissolved portions from their original amounts; that from the precipitate (Z) were recovered 1.851 g. or 46.27 per cent. the original hydrous silicate of zinc. In other words, no change has taken place in the hydrous silicate of zinc, so far as its amount and chemical combination are concerned, whereas the amount of zinc carbonate (its original weight is 4 g.) left in the residue X is only 1.82 g. or 45.5 per cent. of the original amount. Its remainder (theoretically 2.18 g. or 54.5 per cent.) has undoubtedly contributed to the precipitation of the meta-silicate of zinc, the precipitate Y, as will be shown in the following:

	Grams
The original weight of $ZnCO_3$	4.000
The amount left in residue X	1.8200
The amount lost	2.1800

According to its molecular weight, 2.18 g. $ZnCO_3$ contain 1.137 g. metallic zinc, and if this theoretical amount had all been precipitated out as zinc meta-silicate, there would be 2.464 g. $ZnSiO_3$ as against the actual value of 2.376 g. (2.3763 − 0.0003 = 2.376). The difference between the two is about 3.5 per cent., which is evidently lost in the course of manipulation.

From these results, it is clear that zinc carbonate may be replaced by a solution of soluble silicate to form zinc silicate, but empirically, this compound in no way resembles calamine, since the water is not present in sufficient amount. The water, in the case of true calamine, would constitute one-half of the molecular proportion of the oxide of zinc as required by the formula, $2ZnO.SiO_2.H_2O$.

It is also, however, a recognized fact that the water contained in calamine is basic and the mineral is probably a basic meta-silicate of zinc. For this reason, its formula may be represented by $Zn(OH)_2.ZnSiO_3$ in place of $2ZnO.SiO_2.H_2O$. (See the experiment under 6.)

6. *The Reactions of a Mixture of Hydroxide and Silicate of Sodium with a Dilute Solution of Zinc Sulphate.*—From the above suggestions it may be readily inferred that a mixture of Na_2SiO_3 and NaOH will precipitate some such compound, partly as zinc hydroxide, and partly as meta-silicate, from a dilute solution of a zinc salt. The following experiment

was designed in an attempt to demonstrate these suggestive properties of the mineral under discussion.

Experiment 11.—2,000 c.c. of very dilute solution of $ZnSO_4$ (1.61 g. $ZnSO_4$ per liter) were treated with an equally dilute mixture of Na_2SiO_3 and NaOH in excess. A rather heavy flocculent precipitate gradually collected. The amount obtained was 2.412 g. and yielded:

Chemical Components	Per Cent.	Mineral Components	Per Cent.
H_2O	7.463	$ZnSiO_3$	58.83
SiO_2	25.020	$Zn(OH)_2$	41.21
Zn	54.180 = 67.47 ZnO		
	99.953		99.95

According to this analysis, the amount of zinc in the precipitate was 1.307 g., as against 1.308 g. originally present in 2,000 c.c. of $ZnSO_4$ solution. By recasting from its chemical components, it is found that there are 1.4174 g. as zinc meta-silicate ($ZnSiO_3$) and 0.994 g. as zinc hydroxide or 58.83 per cent. $ZnSiO_3$ and 41.21 per cent. $Zn(OH)_2$. If we consider the precipitate obtained in this way as a single compound, the writer sees no objection to writing $ZnSiO_3.Zn(OH)_2$ for the chemical composition of calamine as a makeup of one part hydroxide and one part meta-silicate, since neither of the formulæ, $2ZnO.SiO_2.H_2O$ and $(ZnOH)_2SiO_3$, given in Dana's *Mineralogy*, has any greater claims to being representative. It is far from safe, however, to regard this amorphous precipitate, as true calamine, for the reason that, with the exception of its index of refraction (very close to 1.62) no optical confirmations could be obtained from it.

In spite of this lack of sufficient optical evidences, however, there is reason to believe that calamine in nature is probably built up by steps as meta-silicate and hydroxide in their incipient stages, through some natural influence or condition, producing a continuous alternation in the formation of this mineral, which we are not able to imitate in the chemical laboratory. Since even a very dilute solution of soluble silicate (Na_2SiO_3) and zinc salts always gives an amorphous precipitate at the ordinary temperature and pressure, and since amorphous calamine is not encountered in nature, the writer is constrained to believe that the natural calamine must have crystallized very slowly from very dilute siliceous solutions.

D. *High Temperature Methods*

7a. *The Influence of Temperature on the Formation of Calamine.*—*Experiment* 12: In connection with the preparation[35] of willemite according to M. Gorgeu's method,[36] the writer succeeded in obtaining minute crystals of calamine instead, by treating a mixture containing one part of H_4SiO_4 and 30 parts of one equivalent of Na_2SO_4 and one-half equivalent of $ZnSO_4$. In these experiments only anhydrous Na_2SO_4 and $ZnSO_4$ were used. Various amounts (from one-half to one equivalent) of the latter compound in different experiments were tried, but one-half equivalent of $ZnSO_4$ with one equivalent of Na_2SO_4 in 30 parts of these two compounds, is the suitable amount. All three compounds, well mixed in a porcelain crucible made for high-temperature work, were first heated over a Bunsen burner; and when the fusion began, the crucible was transferred to the "asbestos nest" built in an iron vessel, which was placed in the middle of an assay furnace previously heated. By so doing the temperature inside of the "asbestos nest" may be easily kept at or below 900°C. (this was measured by a LeChatelier thermo-couple pyrometer, protected by a long porcelain tube, which was inserted vertically into the "nest") in order to prevent any zinc from volatilization during the fusing, while the temperature outside the iron vessel is probably over 1,000°C. The heating in this was continued from one-half to one and one-half hours. If the porcelain crucible is transferred into the furnace without using the "asbestos nest," the results are comparatively poor and indicate the volatilization of zinc by the sudden increase of temperature and quick fusion. In this case the odor of SO_2 is quite noticeable near the furnace. On the other hand, it has been experimentally shown that, when the fusion is quite complete, if the "asbestos nest" is used, the temperature may be raised to 1,300°C. or 1,400°C. without noticeable volatilization of zinc.

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[35] The writer desires to thank R. J. Moore of the Department of Chemistry at Columbia University, most heartily for placing at his disposal the apparatus without which it would have been impossible to conduct satisfactorily any of these experiments, and to C. J. Morrell for kind assistance.

After the melt had been kept in the furnace for half an hour or more, the fused mass, which usually became covered with a thin yellowish and sometimes greenish sheet, was placed directly in boiling water. This boiling water treatment causes solidification and usually gives a green surface to the solid mass, especially along the edge. If the mass has been heated too long in the furnace, the solution from the boiling water usually shows a basic reaction; if under-heated the reaction with litmus paper is neutral. By heating in the furnace from half to one hour in the "asbestos nest" as described above, we secure the best results.

After the fused mass, solidified in the boiling water, had been cooled, the contents were taken out with a knife and boiled in distilled water for a long time in order to remove any sulphates crystallized out on the sudden cooling. The desired silicate, if formed, would remain insoluble either in masses or in crystals. This is found to be true, but in each case the crystals of the silicate, as proved below, are very minute and are so small in amount that it is difficult to separate them from the greenish

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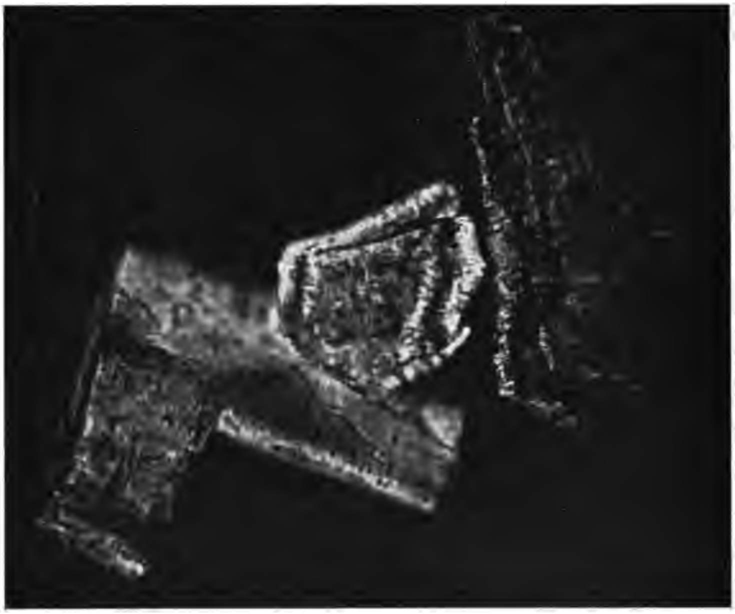
[36] *Bulletin* 10, *Sociètè, Francaise de Mineralogie*, pp. 30 to 39.



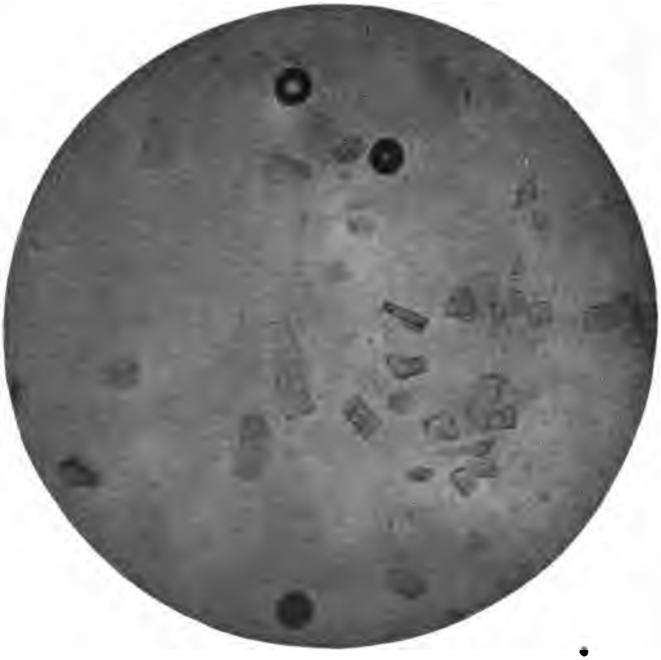


FIG. 4.—MICROPHOTOGRAPHS OF THE CRUSHED FRAGMENTS FROM THE SYNTHETIC CRYSTALS OF CALAMINE PRODUCED BY HEAT FROM A CHEMICAL MIXTURE.

No. 1, magnified 215 diameters, crossed nicols. No. 2, magnified 25 diameters, plain light.

fine residues which equally remain insoluble and which, by qualitative tests, gave SiO_2, Zn, and water vapor.

These minute crystals of the silicate are different in shape and may be divided into two kinds, respectively white and light green in color. The white ones are more or less tabular in form and fibrous in structure, while the green ones seldom show any definite crystal outline. There is no difference in their chemical composition, but optically the white silicate is positive, whereas the green is negative though both give biaxial interference figures.

As suggested by Mr. Colony, Professor of Mineralogy and Geology in Cooper Union in the City of New York, the white crystals which are optically positive and biaxial may be regarded as *m* calamine; the green ones which are optically negative and biaxial, may be considered as *n* calamine. The latter probably represents a transitional stage in the formation of calamine proper. It is further thought that such variation is not uncommon in the behavior of many minerals. Experience shows that a mineral may under special conditions, change even its system as in case of leucite[37] which cannot remain isometric below 560°C.

The following are the results obtained from the microscopic examinations (see Fig. 4):

	White Crystal	Green Crystal
Shape	Elongated	Lath shaped and elongated
Color (transmitted light)	Light bluish	Light greenish
Indices of refraction	1.63	1.61
Extinction	Parallel	Parallel
Extinction angles	85°–90°	Mostly 90°
Birefringence	<0.04	>0.06
Elongation	+	–
Interference figure	Biaxial	Biaxial
Optical sign	+	–

The following are the chemical analyses of the respective silicates:

	White				Green		
	Per Cent.	Molecular Weight	Molecular Ratio		Per Cent.	Molecular Weight	Molecular Ratio
H_2O	7.457	18.0	0.4143	H_2O	6.825	18.0	0.3792
SiO_2	24.990	60.4	0.4137	SiO_2	25.030	60.4	0.4144
ZnO	67.480	81.4	0.8289	ZnO	67.503	81.4	0.8292

The molecular proportion of water for the green silicates is about 0.82 per cent. lower than what is theoretically required, but this difference is so small that it may be safely attributed to the manipulation. Aside from this, either silicate has the following relation, viz. $H_2O = 1$, $SiO_2 = 1$, $ZnO = 2$. In other words these silicates or crystals have the formula $2ZnO.SiO_2.H_2O$, or $(ZnOH)_2SiO_3$ or $Zn(OH_2).ZnSiO_3$ (as suggested by the writer in an earlier paragraph). The question, which is the correct formula, remains to be determined, and is hardly within the province of this work.

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[37] N. H. and A. N. Winchell. *Elements of Optical Mineralogy*, p. 275 (1909).

From these chemical and microscopic results it is clear that the crystals obtained under the conditions described above, are hydrous silicate of zinc or mineralogically a calamine instead of M. Gorgeu's willemite which is an anhydrous silicate of the same metal. The water content[38] of the crystals, under our chemical examination, did not pass off until heated above the Meeker burner with a strong flame. From these experiments the writer concludes: (1) that M. Gorgeu probably mistook the hydrous silicate for the anhydrous silicate of zinc by neglecting its water content (of which he made no mention in the chemical analysis of his product,

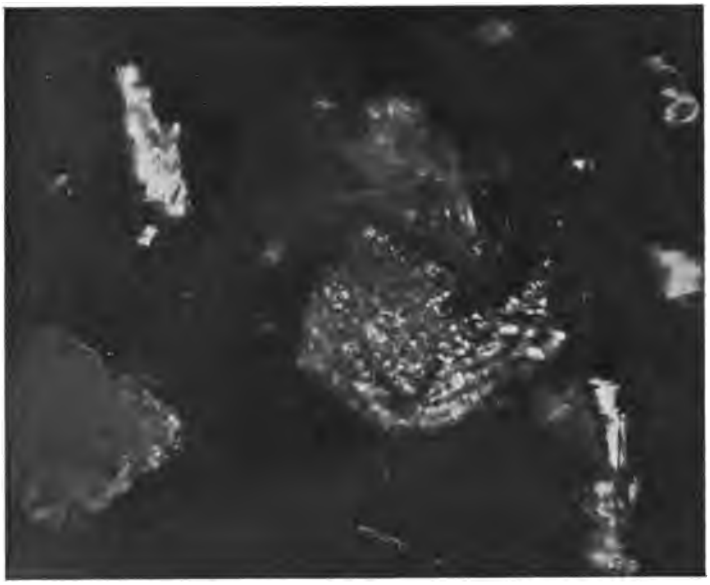

FIG. 5.—MICROPHOTOGRAPH OF THE CRUSHED FRAGMENTS FROM THE FUSED NATURAL CALAMINE.

These are some of the fragments showing the tendency of rearranging their molecules. Under high power some of the very minute hexagonal crystals in the fragments are well shown. This is no doubt due to the influence of heat, but these newly rearranged crystals are too minute to show on photograph. Magnified 215 diameters. Cross nicols.

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[38] According to Fock, "the water goes off only at a red heat, the mineral being unchanged at 340°C. Dana. *System of Mineralogy*, Sixth Edition, p. 548.

and his optical description is rather insufficient); and (2) that he probably took natural calamine instead of willemite (in some instances these two minerals resemble each other and are difficult to distinguish, especially when they are wholly white, and exhibit similar radial structure—as indeed is rare, yet occurs in some of the white willemite from Franklin Furnace, N. J.) for comparison with his synthetic product, since he declared that the elongation of his crystals was parallel to *c*—which is one of the common characteristics of most of the fragments from the crystals obtained by the writer, and which is very common with natural calamine as well as willemite in thin sections. It, therefore, cannot be used as a criterion for the determination of either mineral.

In one respect this synthetic work on calamine may seem to some of the geologists, to be inadequate, namely the temperature and the boiling-water treatment of formation. Field observation points to ordinary temperature. Doubtless, it would be highly desirable to obtain similar crystals of calamine at ordinary temperatures. It must be remembered, however, as pointed out some time ago by well-known geo-chemists, "that there is nothing in our knowledge of the minerals to indicate anything more than continuous changes in the necessary conditions as the temperature falls." It must also be admitted that to imitate a process in nature will be always more or less a matter of extrapolation.

7b. *Possible Effect of Heat on Calamine and Smithsonite.—Experiment* 13.—It is known that artificial zincite and willemite occur in furnace slags and in iron and lead furnaces from the ores carrying zinc. It has been suggested by Dr. Kemp that the ores of zinc at Franklin Furnace and Ogdensburg, N. J., are "the result of metamorphism." This suggestion led the writer to investigate the effect of heat on calamine and smithsonite. The experiments described below are of very simple character, consisting of only four steps:

1. Fusion of the Natural Specimens of Calamine and Smithsonite.—Mere heating in a good gas furnace, does not produce any result beyond driving off H_2O from calamine and CO_2 from smithsonite. The resultant products are usually crumbly and brittle, although they still retain the original structures. No optical change was observed. It is worthy to note, however, that when water is driven off from calamine by heating, the product shows the fluorescent property which is commonly used in testing willemite at Franklin Furnace, N. J. The writer obtained the fluorescence by burning "magnesium ribbons" before the sample.

The results of fusion are quite different. The H-O flame was employed for the fusion, and the fused mass was cooled slowly in each case in the neighborhood of the flame. Fragments of the fused portions of the respective minerals were mounted on slides.

2. Examination under the Microscope.—The fused fragments from smithsonite gave uniaxial positive properties indicating a change in op-

tical sign, since for smithsonite, the optical sign is negative. (Out of hundreds of fragments only a few showed this change the rest were overfused and became cloudy under the microscope.) With the fused calamine, some difficulties were experienced, but the writer succeeded in finding some fragments showing uniaxial positive properties, indicating that the mineral had changed its interference figure; since for calamine, the interference figure is biaxial. It was also observed under the microscope in some fragments that the calamine was completely changed being fully composed of minute six-sided crystals. This seems to indicate the molecular change by recrystallization to willemite.

3. Chemical Analysis.—These analyses do not demonstrate that the fused specimens are molecularly changed; they merely indicate the H_2O and CO_2 are absent after the heat treatment.

	Fused Calamine			Fused Smithsonite		
	Per Cent.	Molecular Weight	Molecular Ratio	Molecular Proportion		Per Cent.
SiO_2	27.06	60.4	0.4584	1	ZnO	100.18
ZnO	72.93	81.4	0.8960	2	CO_2	None
H_2O	None					

4. Test for Fluorescence.—The fused portion from calamine shows fluorescent color, though less strongly than the willemite from Franklin Furnace, N. J.

These results strongly suggest that under the application of both heat and pressure, as they commonly exercised their influence under metamorphic conditions, the still more striking and definite changes might be exhibited. But this cannot be obtained without the necessary equipment for both pressure and temperature. A system of slow cooling is also essential.

E. *Summary of the Experimental Work*

The oxidation and solution of blende is prerequisite to the formation of any one of the oxidized ores, and the process of oxidation is brought about by reactions of the sulphide with a ferric iron salt. Ferric sulphate is most conveniently used.

When the iron salt is ferric sulphate the zinc sulphide is usually converted into sulphate; but the solvent action of this salt does not seem to be in the proportion to its concentration, as only a small percentage of the ferric is reduced to ferrous by the oxidation of the zinc sulphide. These results suggest that the chemical equation, $Fe_2(SO_4)_3 + ZnS = FeSO_4 + ZnSO_4 + S$, represents only the chemical reaction and not the chemical ratio.

When a ferrous salt is alone present (as it rarely is, in quantity, in the zone of oxidation), no oxidation of zinc sulphide can take place unless free air is accessible; and when air is introduced, the sulphide may be converted into sulphate by the oxidation products of the ferrous salt prior to the formation of any one of the oxidized ores of zinc.

Soluble salts of such metals as Ag, Cu, and Pb in solution, exert a powerful solvent action on blende. Of these, a solution of silver salt is much more powerful than a copper salt solution and the latter is more so than a salt solution of lead. Lower and different concentrations of the sulphates of silver and copper were used. In case of lead, lead nitrate was substituted.

A solution of cupric sulphate has a powerful solvent action on the natural sulphide of zinc, hence in an orebody containing much copper sulphide or galena or both, little sphalerite may be expected in the horizons where the former two sulphides are rich, as most of the blende will be taken into solution and the metal will be redeposited somewhere as one of the oxidation products; and conversely, if rich zinc sulphide be found in a certain horizon, we may be pretty sure that the orebody cannot have contained much lead (or copper) sulphide, and that little oxidation products should be expected. This accords with the fact that zinc carbonate and zinc sulphide have been found in different horizons, as if they had no relation to each other, as in the Circle Mines at Oronogo, Mo.

When a solution of an alkali or calcium bicarbonate is mixed with a solution of zinc sulphate or chloride, a zinc carbonate is precipitated; but when a reagent containing normal carbonate, such as Na_2CO_3 or K_2CO_3 is used, only a basic carbonate or hydrozincite is obtained.

When a dilute silicate solution (Na_2SiO_3) is mixed with a dilute solution of zinc salt, only a precipitate of a meta-silicate of zinc is obtained. In the presence of carbon dioxide, this silicate solution replaces also zinc carbonate, to form meta-silicate of zinc. If the reagent contains a mixture of sodium silicate and sodium hydroxide a "complex" precipitate of meta-silicate and hydroxide of zinc may be obtained. In chemical composition this precipitate corresponds to natural calamine, but by reason of its amorphous character no definite confirmation is here offered.

The presence of carbon dioxide exerts a powerful solvent action on all the oxidized ores under consideration. Of these, zinc carbonates are more soluble than the basic silicate of zinc in waters containing an excess of carbon dioxide. This may account for the fact that no carbonic acid, as far as we know, has ever been detected in the analysis of the waters from the mines of these oxidized ores. Upon release of the carbon dioxide any of them may be recovered.

Limestones immersed in dilute solutions of zinc sulphate were found to be replaced by zinc carbonate; but the replacement is much more rapid with the impure limestone than with the pure ones. This result suggests that the presence of organic matter in the impure limestone has probably exerted some influence in the rate of replacement.

At or a little below 900°C., minute crystals of calamine may be prepared from a chemical mixture containing definite proportions of H_4SiO_4, Na_2SO_4(dried), and $ZnSO_4$ (dried). Both the chemical analysis and the optical characters of the crushed fragments all point to their identity with natural calamine.

The influence of heat on calamine and smithsonite results in the change of their optical characters into those of willemite and zincite; but the optical tests were so difficult to carry out satisfactorily that it appears doubtful whether the anhydrous silicate and the oxide of zinc can be formed in this way.

III. Petrographic Work

Introductory

Petrographic data concerning the mode of formation of the ores of zinc are, so far as we are aware, limited to descriptions of occurrences of blende.[39] As to the oxidized ores practically no attempt[40] has heretofore been made to discuss the paragenesis of the mode of their formation in the light of their petrology. They have naturally been regarded as products from the alteration of sphalerite, but the conclusion was wholly drawn from field studies.

The data given below are the results of a study of thin sections from ore specimens, chiefly selected from the collection of ores of the Department of Geology at Columbia University, and representing only a few localities chosen for this purpose.

This was undertaken in conjunction with the experimental work, in order to arrive at some definite conclusion as to (a) the genetic order of the precipitation of zinc carbonate and zinc hydrous silicate; and (b) the influence of inclosing rocks and associated minerals, principally limestone and pyrite respectively, upon sphalerite and its oxidation products. For this reason the descriptions are not necessarily confined to the oxidized ores alone but also include the sulphide; and the specimens from which thin sections were cut are grouped, not according to their localities but according to their present form of occurrence, into two types as follows: (1) Sphalerite occurring in limestones or cherts from southwest Wisconsin; southwest Missouri; Mascot Mine, Tennessee; northwest Arkansas; Wythe County, Virginia; and Saucon Valley, Pennsylvania; and (2) partially altered sphalerite and the oxidation products, calamine and smithsonite, from southwest Missouri, northern Arkansas; and Leadville, Col.

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[39] C. E. Siebenthal. *Mineral Resources*, 1911, p. 353 et seq., U. S. Geological Survey, W. Lindgren, *Mineral Deposits*, p. 422 (1913); and T. L. Watson, *Bulletin No.* 1; *Geological Survey of Va.*, pp. 95 to 96 (1905).

[40] Watson. *Idem*, p. 45.

1. *The Occurrence of Sphalerite in Limestones or Cherts*

Different views have been advanced by numerous authors as to the mode of deposition of zinc sulphide in limestones and cherts or in their breccias. Later writers have generally agreed, however, that the ores of this type were first derived from above by the decomposition and erosion of superimposed strata, and finally concentrated by meteoric waters in brecciated limestones or cherts.

The result of the present investigation on thin sections from the sulphide ores of zinc points to some such theory. The sections examined show that in many instances the zinc sulphide has filled completely the intersticial space in the brecciated country rocks, principally limestones and cherts. In others, the sulphide appears to have impregnated the country rocks and filled the cavities in the limestones as if it had been deposited with the inclosing rock.

Among the associated minerals are galena, pyrite, magnetite, limonite, secondary quartz, and calcite, flourite and greenockite. The latter two were not observed in the sections here examined; and galena was absent, or occurred only in minor amount, where sphalerite was dominant. Sphalerite usually incloses galena, indicating the previous deposition of the latter. Of the three iron compounds, pyrite is the commonest, while magnetite infrequently concentrated around the peripheries of sphalerite, and limonite occurs sparingly along cleavage cracks, when such occur. The order of origin of these minerals with respect to the sulphide ore of zinc has therefore appeared to be: (1) Galena (if present); (2) sphalerite; (3) iron sulphides; and (4) secondary quartz and calcite. (2) and (3) may have taken place almost simultaneously and (3) and (4) reverse in some instances. Wherever quartz occurs in these sections, it is invariably of secondary origin.

A specimen from southwest Wisconsin consisted of sphalerite and a siliceous mass on the top, which was thought to be the alteration product, calamine, but upon examination under the microscope was found to consist of aggregates of secondary quartz. Some are cryptocrystalline and others are much larger. In it secondary calcite occasionally occurs (Fig. 6).

The essential relation of secondary quartz to sphalerite appears to be the same in all the specimens studied from the localities indicated above, as shown in Figs. 6 to 11. Irrespective of their localities, they all show that the voids or interstices in the limestone were completely filled with secondary quartz, and that where spaces in the country rock were not wholly occupied by sphalerite the remainder was filled with quartz.

Secondary calcite is occasionally seen in cleavage cracks of sphalerite. These relations indicate that the precipitation and concentration of sphalerite preceded the deposition of secondary quartz and calcite.

Some sections show pyrite penetrating the blende, as well as the calcite and quartz, along their strain- and cleavage-cracks. This means that subsequent to the deposition of most of the sphalerite and of the secondary quartz in the fractured limestone, there came a period of infiltration of solutions, bearing iron sulphide. In these cleavage cracks of sphalerite and quartz, pyrite has been deposited. It further favors the contact of sphalerite and is especially prominent around the edges of the zinc sulphide. The contacts of pyrite and sphalerite thus produced afford the most important factor in the oxidation of sphalerite, discussed in the earlier part of this paper.

Sphalerite is, as a rule, most abundant in brecciated limestone and in the secondary black chert. The latter is largely made up of cryptocrystalline quartz with grains of blende and crystals of dolomite (Fig. 7). The dark color of the chert is due to the presence of much organic matter. In the bedded limestones and white cherts, sphalerite is practically absent.

Summary of the Blende in Limestones and in Cherts.—From the thin sections examined here it may be briefly stated that the deposition of secondary quartz was subsequent to the crystallization of sphalerite. This secondary quartz was probably in part introduced by percolating waters and in part was furnished by the decomposition of cherty limestone and cherts with minor amounts of secondary calcite. The precipitation of pyrite in some cases was almost simultaneous with the crystallization of sphalerite, but it has not had the opportunity since its deposition to exert influence on the oxidation of sphalerite.

2. *The Partially Altered Sphalerite and the Oxidation Products*

The country rock in which the ores of this second type occur is chiefly dolomitic limestone. The associated minerals are galena, pyrite, limonite, hematite (both derived from pyrite), secondary quartz and calcite, and some primary quartz. In no case are they all present in a single section; and primary quartz and pyrite were observed only in specimens from Joplin district, Missouri, and Silver Hollow Mine, Arkansas.

a. *Smithsonite with Blende* (Fig. 18).—This shows the peculiar occurrence of galena around peripheries of residual grains of sphalerite. In some cases galena forms a succession of rings. The sphalerite within a single ring is usually unaffected. The iron oxide associated with smithsonite is usually in the form of limonite. It penetrated or concentrated along the cleavage cracks of smithsonite, as shown in Fig. 18. A characteristic feature of the smithsonite is the occurrence of minute secondary quartz and of some straight and elongated crystals of calamine intergrown in it, as shown in Fig. 14. It is evident that the intergrowth or interlocking of these oxidation products (Figs. 13 and 14) represents the overlapping periods of formation of both minerals. Their amount of each necessarily depended upon the amount of the required substances in the reagents in original sections.

The ores from Silver Hollow Mine, Arkansas, contain a large number of almost round grains of quartz, some of which were completely surrounded by pyrite and others by both pyrite and secondary calcite. These quartz grains are primary in origin, and are believed to have come from the disintegration of some original sandstone, before the penetration of pyrite. The edges of these quartz grains are more or less corroded and in some cases secondary calcite has filled these spots (Fig. 15). The optical examination of this ore indicates that smithsonite is rarely derived direct from sphalerite, but mostly from solutions, after the sphalerite had been oxidized into a soluble salt presumably a sulphate, by meteoric waters. Fig. 16, from the same section, shows sphalerite cut by a veinlet of smithsonite. In some places the section also shows the sharp contacts of smithsonite with sphalerite. Had the smithsonite been formed by direct alteration from sphalerite such a relation could hardly exist. These sections further indicate that simultaneous with or subsequent to the solution of sphalerite in some parts of the ore there occurred a fracturing of the unaltered sphalerite, which permitted the infiltration of solutions bearing smithsonite. In the cracks thus produced in the unaltered portions of sphalerite, smithsonite has been deposited. Subsequent to its deposition is the deposition of secondary quartz which filled the rest of the vein. Pyrite which penetrated sphalerite and calcite along their cleavage weakness in the specimen is only observed in part in thin sections.

b. *Calamine with Blende.*—Where blende was not entirely altered, its periphery was surrounded by a ring of galena (see Fig. 18) and occasionally by limonite. Like smithsonite, calamine rarely replaced sphalerite in place. Its crystals usually exhibit a habit of radial or of massive structure. This habit of structure (Figs. 13 and 17) suggests at least one thing, namely, that its crystallization in general took place from solutions, and not by direct alteration of sphalerite.

The rich ore of calamine may also carry some smithsonite as a minor product, and the same is also true in the reverse case. Both of them may replace calcite as the result of metasomatic replacement, as shown on Fig. 18. It is sometimes difficult to distinguish smithsonite from calcite, especially when they are closely associated. The difference in their relief generally affords a clue to their distinction. In nearly all cases, the surface of smithsonite appears very shaggy under the microscope.

c. *Smithsonite and Calamine.*—The sections here examined were cut from the ores containing both calamine and smithsonite and are practically free from gangue. In some instances they are intergrown in each other; in others they are interlocked with each other, as already described above.

As to their structure, each of these two minerals seems to have followed a definite habit of crystallization. With calamine the radial structure is common, whereas smithsonite is usually granular, especially in high-grade ore (Figs. 13, 14, 17).

A specimen, known to contain both calamine and smithsonite, from Leadville, Col., was found to be mostly calamine, largely made up of crypto-crystalline and elongated crystals, as shown in Fig. 19.

Fig. 20 from the same section shows narrow strips of iron oxides (hematite and limonite) surrounded by calamine. These oxides appear to have concentrated along the partings and the cleavage cracks of calamine. Where this mineral is in the form of minute crystals, the surface is irregularly coated also with iron oxides (Fig. 19), chiefly limonite. The iron oxide was undoubtedly an iron sulphide in origin, and was later reconcentrated and reprecipitated in its present form, after its oxidation product, generally ferric sulphate, had been destroyed in the oxidation of sphalerite, with which it was in contact in the original rock.[41]

Summary on the Partially Altered Sphalerite and its Oxidation Products. —From the examination of these sections it may be said that:

(a) When sphalerite was surrounded by a ring of galena or iron oxide, its further oxidation was terminated.

(b) Sphalerite was rarely replaced by its oxidation products in place. In most cases the solution of sphalerite, through the reaction of the oxidation products of iron sulphides, preceded the transportation or migration, and the deposition of the metal in the form of carbonate or hydrous-silicate.

(c) The alteration of sphalerite proceeded inward zonally from the exterior.

(d) No definite order of the formation of calamine and smithsonite was observed. They may intergrow with each other or either may precede the other. Thus the formation of these oxidation products depended upon the availability of the required substances in the reagents which had come into contact with a solution containing the metal zinc, presumably as zinc sulphate.

(e) Both calamine and smithsonite replaced calcite, or the limestone might pass into these products in orebodies. This relation has also been observed in some zinc mines.[42]

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[41] T. L. Watson. *Bulletin No.* 1, *Geological Survey of Va.*, p. 45 (1905).

[42] U. S. Grant. Report on the Lead and Zinc Deposits of Wisconsin, *Bulletin No.* 14, *Wisconsin Geological and Natural History Survey*, p. 59 (1906).

IV. Conclusions

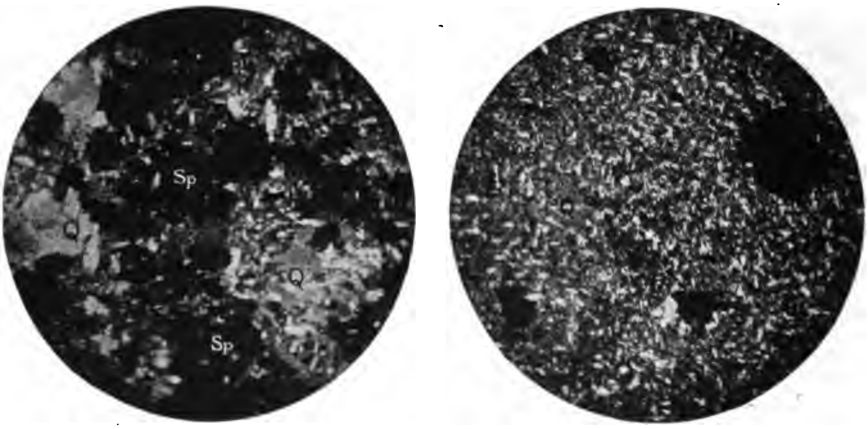
From the chemical and petrographic data presented in this paper we have sufficient evidence to justify the following conclusions:

1. Sphalerite is first attacked by the oxidation products of iron sulphides before the formation of the secondary minerals containing zinc.
2. The formation of the oxidized ores rarely occurs by direct replacement of sphalerite.
3. Normal zinc carbonate (smithsonite) is formed when the reagents in solutions are bicarbonates; and basic zinc carbonate (hydrozincite) is formed when the reagents are normal carbonates.
4. Limestones may be replaced in nature by solutions of a zinc salt, usually the sulphate.
5. In nature both smithsonite and calamine replace limestones.
6. No free carbon dioxide can be present where silicates and carbonates of zinc occur, and conversely, these minerals of zinc do not appear where free carbon dioxide is present.
7. "Amorphous calamine" may be formed from solutions of silicate and hydroxide under ordinary conditions.
8. Calamine in nature is probably formed by steps, and is precipitated from waters containing ions of some gelatinous silicate and of hydroxides.
9. It is evident that calamine is formed under ordinary conditions in nature, but time is a great factor in the process. Until this factor is satisfied we can hardly expect the production of crystals of calamine by wet methods.
10. Minute crystals of calamine may be formed by heat at or below 900°C. from a chemical mixture, as set forth under Experiment 12.
11. No confirmation is here offered of the change of calamine and smithsonite to willemite and zincite by heat, other than that heat produces changes of optical characters into those of willemite and zincite respectively.

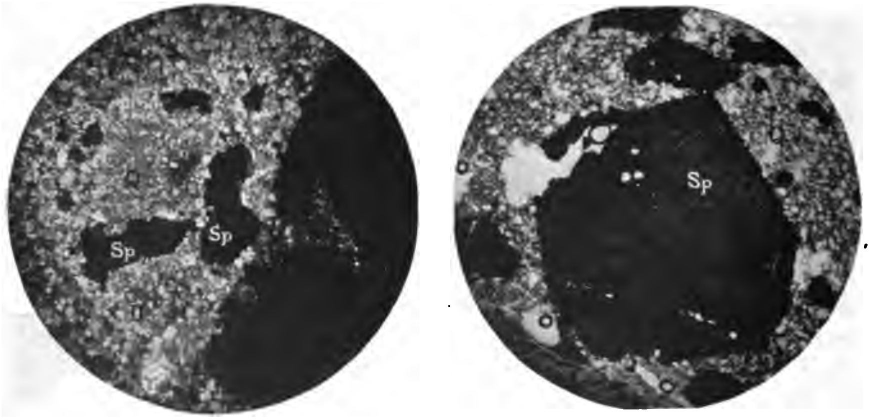
V. Microphotographs and Photographs with their Explanations

Abbreviations

C = calamine
Ca = calcite
D = dolomite
G = galena
I = iron oxides
P = pyrite
Q = secondary quartz
PQ = primary quartz
S = smithsonite
Sp = sphalerite



Figs. 6 and 7.—Magnified 17 Diameters; Crossed Nicols.



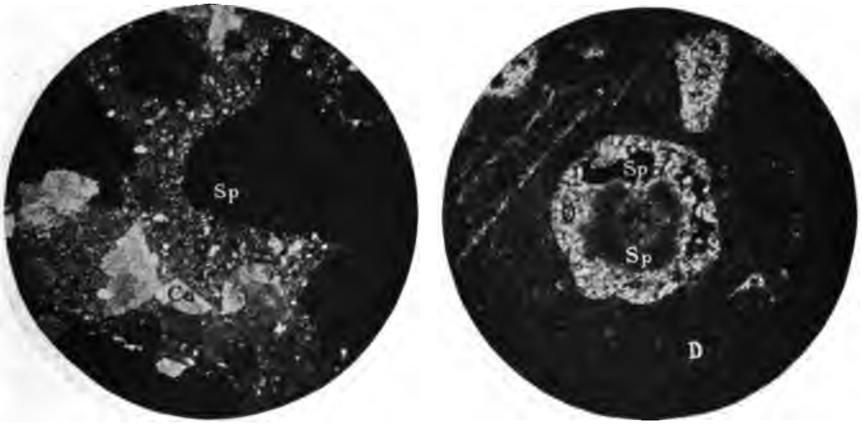
Figs. 8 and 9.—Magnified 17 Diameters; Crossed Nicols.

Fig. 6.—Thin sections of blende ore from Southwest Wisconsin. Black areas represent sphalerite. White areas, secondary quartz, which was formerly thought to be zinc silicate such as calamine.

Fig. 7.—Thin sections of black chert containing ore from Orongo Circle Mine, Orongo, Mo. Black areas, sphalerite. Shows cryptocrystalline character of the secondary quartz in the black chert.

Fig. 8.—From the same locality as 7. Shows the voids in the limestone were entirely filled with both sphalerite and secondary quartz.

Fig. 9.—Thin sections of the sulphide ore from Joplin, Mo. Note the sphalerite was entirely surrounded by secondary quartz.



FIGS. 10 AND 11.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

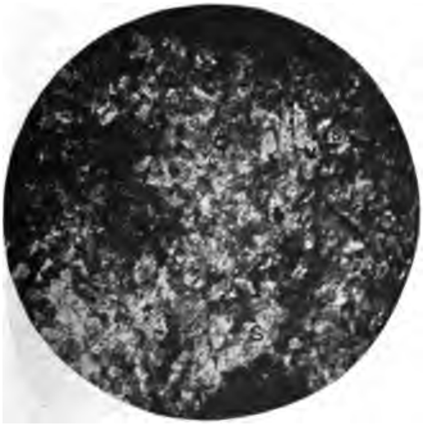


FIG. 12.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

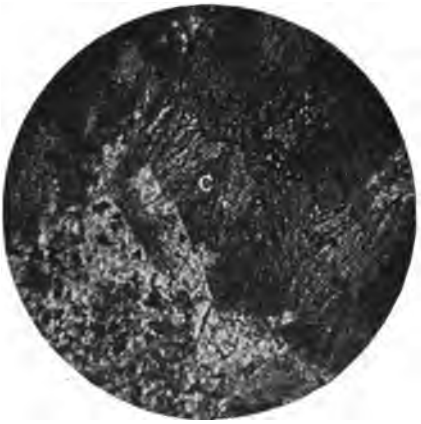


FIG. 13.—MAGNIFIED 140 DIAMETERS; PLAIN LIGHT.

FIG. 10.—Thin section of the ores from Northwest Arkansas. Shows the secondary quartz and the filling of minute cracks accompanied by replacement of the sulphide by the quartz. Re-entrant angles along some margins of the sulphide are well shown.

FIG. 11.—Thin section of the ores from Mascot Mine, Tenn. The dark area within the lighter cicular portion represents sphalerite. A large portion of it was probably removed by a siliceous solution from which the aggregates of secondary quartz deposited around the remnants of the sulphide.

FIG. 12.—Thin sections of the oxidized ores from San Francisco Mine, Missouri. Shows the intergrowth of calamine and smithsonite. The latter constitutes the major portion of the section including the dark areas; while the latter is only in minor amount and mostly in elongated form of minute crystals.

FIG. 13.—From the same locality as 12. Shows the growth of smithsonite in calamine. Note the difference in their structure. Calamine exhibits its radial habit of crystallization, whereas smithsonite is in lath shape.

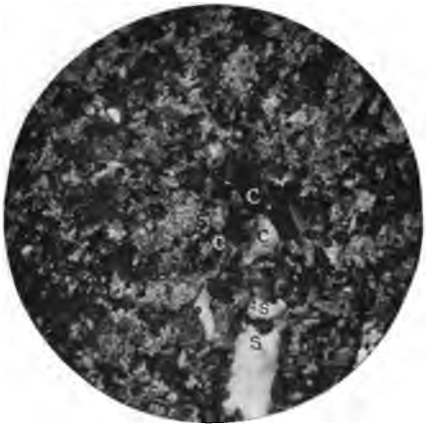


FIG. 14.—MAGNIFIED 17 DIAMETERS; PLAIN LIGHT.

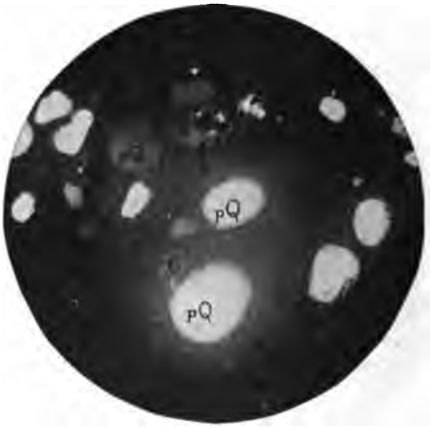
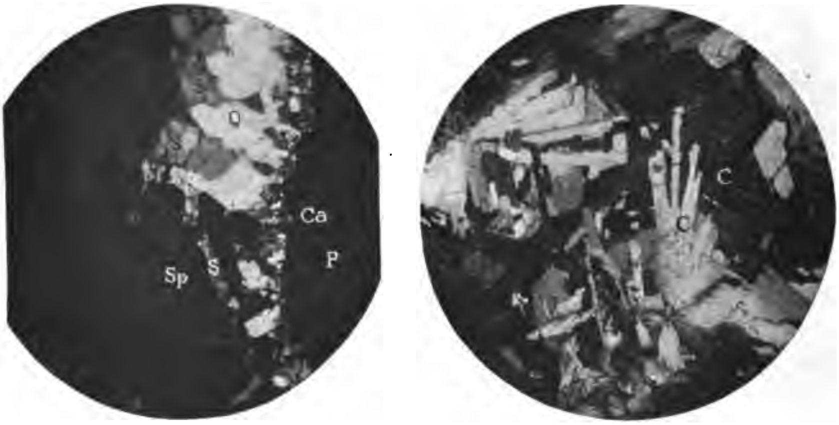


FIG. 15.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.



FIGS. 16 AND 17.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

FIG. 14.—From the same locality as 13. Note the interlocking of calamine and smithsonite. The darker surface of calamine is due to the cloudy appearance of the crystals.

FIG. 15.—Section of the ore containing blende and smithsonite from Silver Hollow mine, Arkansas. Note the round grains of quartz and their corroded edges filled with pyrite and secondary calcite.

FIG. 16.—From the same locality as 15. Note the veinlet of smithsonite in sphalerite and their sharp contacts. In this, the oxidized ores had probably removed only a short distance and almost remained in places, but it nevertheless indicates the fact that the solution of sphalerite preceded the deposition of smithsonite.

FIG. 17.—Thin sections of calamine ore from Joplin, Mo. Note the radial habit of growth in almost all directions.

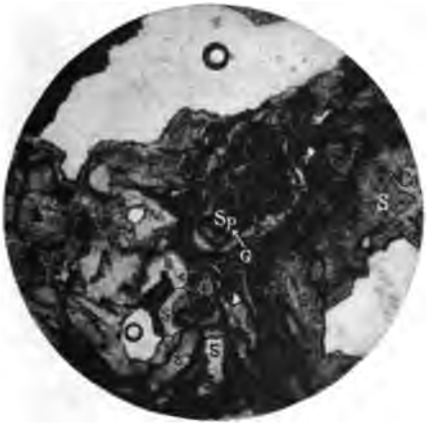


FIG. 18.—MAGNIFIED 17 DIAMETERS; PLAIN LIGHT.

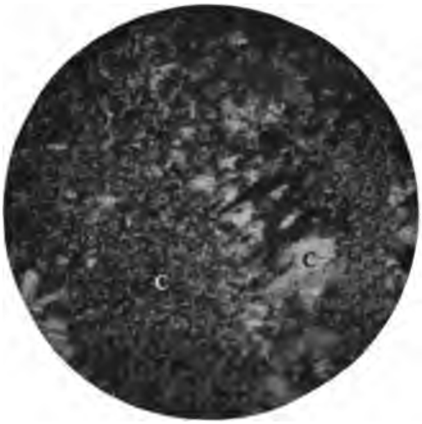


FIG. 19.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

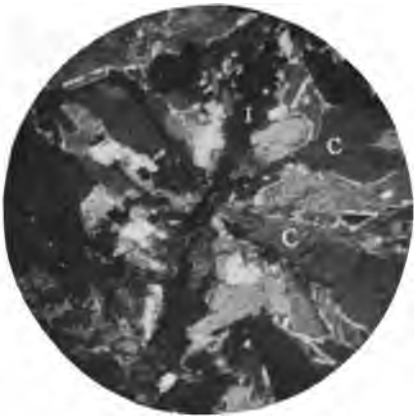
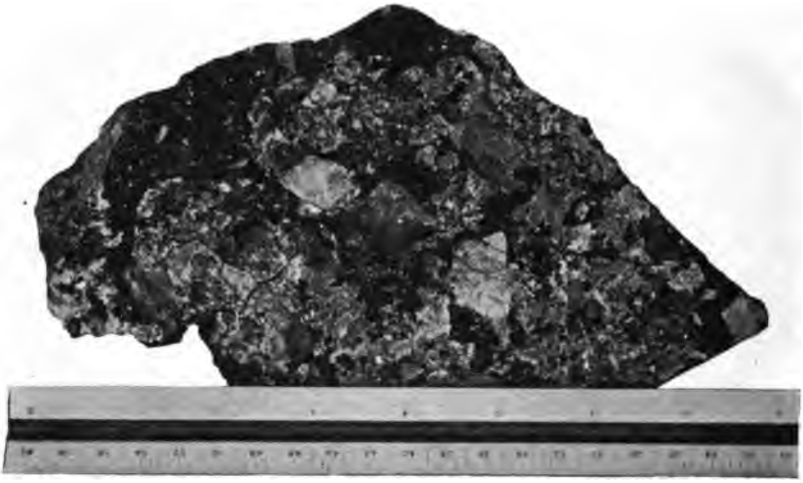


FIG. 20.—MAGNIFIED 17 DIAMETERS; CROSSED NICOLS.

FIG. 18.—Thin sections of the ore specimen from Northwest Arkansas. Shows the galena "rings" in dark color inclosing the residual grains of sphalerite. The areas outside the innermost "rings" were replaced by smithsonite. Note also the replacement of calcite by smithsonite to the right. This probably indicates the fact that upon the solution of sphalerite, a carbonate solution immediately came in and gave rise to the precipitation of zinc carbonate. See discussions under the formation of zinc carbonate in the earlier part of this paper.

FIG. 19.—Thin section of the oxidized ores from Leadville, Col. Note the two types of crystals; one is in an elongated form and the other is mostly cryptocrystalline. Its cryptocrystalline character almost resembles that of some of the secondary quartz in the blende-bearing cherts. (See Microphotograph No. 7)

FIG. 20.—From the same locality as No. 19. Note the narrow strips of iron oxides concentrated along the crystallographic partings and the cleavage cracks of calamine.



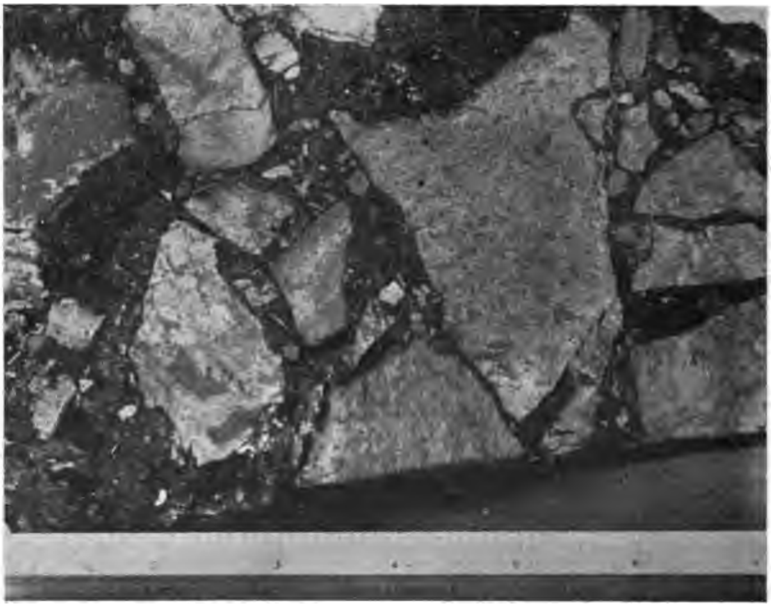


FIG. 21.—Specimen of blende ore from Oronogo Circle Mine, Oronogo, Mo. Brecciated structure of the country rock. Dark areas represent sphalerite and the fine grains of black chert cemented together. The lighter portions represent brecciated limestones and white cherts.

FIG. 22.—From the same locality as No. 21. The white areas represent white cherts in which practically no sphalerite occurs.

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DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, Sept., 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Stresses in the Mine Roof

BY R. DAWSON HALL,* BROOKLYN, N. Y.

(San Francisco Meeting, September, 1915)

THE stresses in the simplest structures are often those we find most difficult to analyze. The most complex condition in mine stresses is found in simple tunnels where the roof, the sides, and the floor are a monolith. The functions of the parts are like the parts themselves not distinct and specialized, and the problems to be solved are like those in a metal structure with riveted joints or a redundancy of bars.

This difficulty explains perhaps why the condition has not been treated. But just because it cannot be discussed in its entirety is no reason why it should be treated as an action of parts with specialized functions as a roof beam with supports and a foundation. The problem cannot be ignored on the ground that it is not of sufficient importance to warrant careful consideration, because conditions of complete monolithism, of which the tunnel is the type, are found materially unchanged in room-and-pillar and in longwall work.

This unity between roof, sides, and floor, which to the coal miner is a difficult conception, really deserves a scientific appellation, and perhaps holoid (from *holos*, whole, and *eidos*, form) will serve the purpose as well as any other.

In a simple tunnel the roof, the sides, and the floor form integral parts of one and the same structure, and the distortion of one cannot be conceived without a consequent strain in the others. Thus when the roof of the tunnel droops by reason of its weight, the upper parts of the sides are drawn in because they are integrally connected with the roof and must approach each other whenever, by the sagging of the roof, the distance between any two points in it is diminished. (See Fig. 1.)

The sides in their turn operate on the floor of the holoid structure, producing a tensile stress. The writer has always been impressed with the value of soap as a means of illustrating the action of mine stresses. With that idea in mind a cake of naphtha soap measuring 4⅜ by 2¼ by

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*Associate Editor, *Coal Age*.

1¾ in. was taken and a tunnel was made through it 1¾ in. long, 2½ in. wide and 1¼ in. high. (See Fig. 2.) A load was then placed at the mid-span of the tunnel. Eventually, the upper bar, or "roof," broke at the center line and along both "ribs" of the tunnel, the breaks being approximately vertical and proceeding, as might be expected, on the ribs from the "surface" downward and at the center line from the tunnel upward, the failure being from bending moment, not shear.

This is interesting because it shows that breakage *at the ribs* is not necessarily evidence of shear. It may be only a demonstration of a holoid structure. It is the form of failure whenever coal is blasted down and not an infrequent form of roof demolition. The test on the soap tunnel further showed that as soon as rupture takes place in the roof of the tunnel, there must inevitably come a thrust on the ribs. The tension draws them together till rupture occurs and then the two roof units, in endeavoring to revolve, crowd each other and push on the opposing walls. (See Fig. 3.)

When a holoid structure, by reason of the weakness of the floor or because of a lack of adhesion between the ribs and the floor, ceases to engage the floor in its movements, then its shape as a structural element roughly resembles the Greek letter π and for want of a better name we might term the new element a pyoid structure. (See Fig. 4.) The doctors use the word for a totally different purpose with little excuse. "Pyonoid" is the word which they should use to express the attributive "pus-like."

By reversing the soap tunnel after the roof has been caved the weakness of the pyoid structure is made clear. As soon as pressure is brought on the new roof (formerly the floor of the tunnel), the two ribs are seen to recede markedly and if the floor were intact this would result in a well-developed stress in that element of the holoid structure. (See Figs. 5 and 6, showing progressive demolition.)

Probably it is well here to express a belief in the importance of the holoid. The general notion is that all the beds shear horizontally along the lines of stratification and that it is a mistake to consider the mine or even the roof as a monolith. It is true that most of the accidents in mines are due to the lack of unity in the roof. What we call draw-slate accidents are almost wholly due to this fact. Nevertheless, it is interesting to note how firmly roof and coal are usually "burned" to one another. Even when undermined and sheared on both sides the coal often fails to fall, being supported by the vertical shearing strength of the one side still attached and by the adhesion to the roof. The writer is hardly prepared to state when the holoid structure ceases to exist, and of course the time and conditions will vary with the materials under consideration.

It is obvious that with the holoid structure, the ribs being drawn

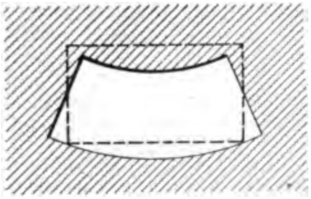
FIG.1

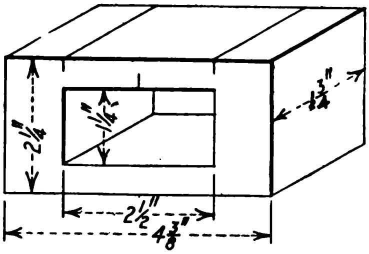

FIG.2

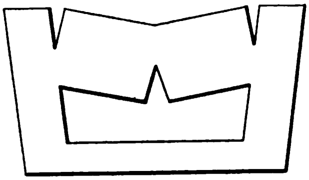
FIG.3

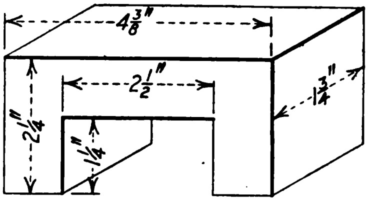

FIG.4

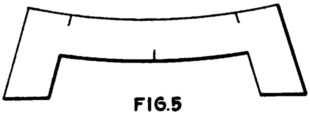
FIG.5

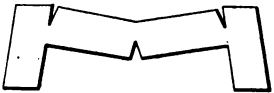
FIG.6

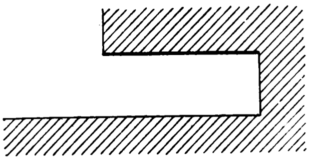
FIG.7

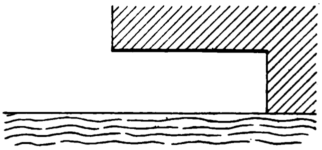
FIG.8

Fig. 1 shows in broken lines a cross-section of a tunnel and in full lines the same tunnel distorted by pressure. This is a typical holoid structure.

Fig. 2 shows a soap model to which reference is made in the article.

Fig. 3 exaggerates the upper or roof bar in the soap model and shows how the collapse of the roof flrces back the sides by the arch action.

Fig. 4 shows the soap pyoid without load.

Fig. 5 shows the same when the bending of the roof bar takes place, thrusting out the ribs, or legs.

Fig. 6 shows a further action when the revolution of the segments in collapse tends to force back the upper parts of the ribs or legs.

Fig. 7 illustrates a semi-holoid structure.

Fig. 8 illustrates a semi-pyoid structure.

together by the movement of the roof, they must tend more or less to be split vertically and in longwall they will then fall down on the advancing undercut. In certain sub-bituminous mines the writer has noted a tendency toward what he thought was a vertical shear parallel to the headings. This developed rapidly after the work was opened, especially at great depth.

Whether this was only a vertical shear seems doubtful. It may have been due to the holoid character of the structure, the ribs receiving no relief by a horizontal shear between ribs and roof. Instead the roof pulled off the edge of the pillar as the former bent under the load. It was interesting to note that these lines of fracture did not coincide with the normal cleavage of the coal.

This rending along vertical planes eventually throws back the real rib lines far into the pillar. Where the draw slate and roof proper leave one another, we have probably a plate structure superposing one which is holoid or pyoid in character, and in the longwall the breaks back of the face which bring coal and roof down together, or which tend so to do if the latter is not duly propped, are failures of the semi-holoid or semi-pyoid and not of the plate structure above. (See Figs. 9 and 10.)

The breaking of even the holoid roof is not necessarily a sudden, unheralded event, such as one might anticipate from a cursory consideration of the problem. It is clear that the action of the moments cannot destroy the roof without revolving in a degree the elements into which the roof is broken, and any revolution inevitably binds these elements against one another so that they are less able to fall. Either a recession of the ribs or a further demolition of the revolving elements must take place or the roof will not fall. One form of demolition which frequently occurs in shallow workings is vertical shear along the cracks already made by the bending-moment stresses. But horizontal shears may make it possible for the roof masses to revolve and yet fit the space they occupied by a counter revolution of the strata past each other. Or again, the whole mass may be broken up by the rubbing of the opposing faces of the elements as they try to fall.

It is this last action which is in evidence in coal brought down by a shot when it is broken considerably in falling, and that vertical shear is not an important cause of the fall of coal is shown by the fact that there is a distinct tendency for the coal to roll away from the side ribs.

It is necessary now to consider the plate structure in which the roof is considered as a vast plate, a monolith in itself but resting without adhesion on its supports. Whenever there is a mined area the roof is depressed, and being elastic it tends to rise on the surrounding supports, resting its weight on the edges of the surrounding ribs of the excavation. This area of quasi-elevation is followed by another area of depression

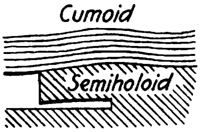

FIG. 9

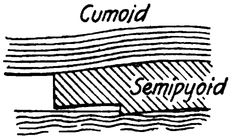

FIG. 10

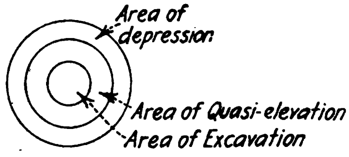

FIG. 11

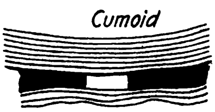

FIG. 12

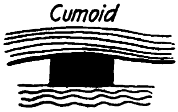

FIG. 13

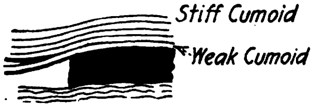

FIG. 14

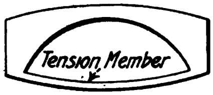

FIG. 15

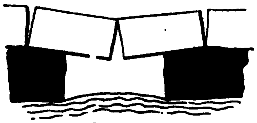

FIG. 16

Fig. 9 shows a semi-holoid such as often occurs in longwall, supporting a cumoid which, being separated from the semi-holoid, has freedom of movement except for the restraint of frictional contact.

Fig. 10 shows a semi-pyoid likewise surcharged by an independent cumoid.

Fig. 11 is a plan of a circular excavation. The sagging of the roof into this area produces an area of quasi-elevation; that is, an area which would be elevated if the mineral itself and the floor surrounding the excavation were incompressible and indestructible. Outside this area is one of depression, the stiffness of the measures lightening the burden on the area of quasi-elevation and loading with increased heaviness the edges of the excavation and the area of depression. It exhibits the fact that the action of the roof is cumoidal, or wavelike, and not confined to the area of excavation. The depressed area is in its turn followed by another area of quasi-elevation, and this again by an area of depression. The size of these waves of course decreases as the area of excavation is left behind.

Fig. 12 shows a cumoid bending over an opening and crushing two pillars.

Fig. 13 shows a cumoid bending over a pillar, the center of which is a point of quasi-elevation. The roof tends to break over such a support.

Fig. 14 shows a weak cumoid, such as a loose draw slate which has broken away from the stiffer cumoid above and probably has insufficient moment of inertia for self support.

Fig. 15 is a Luten bridge, showing the lower tension member, or paving, which ties the abutments together and prevents their recession, thus adding to the strength of the bridge.

Fig. 16 shows a broken cumoid failing to fall owing to an arching action.

surrounding the central depression and the area of quasi-elevation. Thus the roof plate is bent into a series of waves around the central area of disturbance just as the surface of the water is rippled round the point where a stone has fallen and disturbed its equilibrium. (See Fig. 11.)

Of course, the elevations are only relative, not actual, and naturally, like all undulations, as they recede from the point of disturbance they die down. But it is essential to remember that while the holoid structure is to a large extent a closed force chain, this is not nearly so true of the plate, the stresses in which are less localized and circumscribed.

This structure we may dub as cumoid (from *kuma*, a wave). The remarkable feature about such a structure is that it develops points of great stress far away from the disturbing cause, and it may break over the pillar instead of in the opening. The appearance of the Forth Bridge, Scotland, is known to almost every one. The light structural work over the midspan contrasts most forcibly with the heavy structure over the piers. It is the case with all cantilevers and continuous-arch structures, and the roof in the mine is like a continuous arch, only it is continuous not only in one, but in every direction.

It is the peculiarity of the cumoid structure that the stresses it involves may be greater farther from the point of disturbance than at some nearer point. But, like the holoid structure and the pyoid, it puts the greater burden on the pillar's edge. The center of the pillar between two large open spaces may be relieved from much of the normal pressure because of the bending of the cumoid roof over the pillar. (See Fig. 13.)

There is some evidence that in actual operations in some sections of the country the roof soon breaks by horizontal shear into two or more separate cumoid structures, of which of course one is free of external load, while the others, though below other cumoids, may or may not be loaded. If the stiffness of the upper cumoid, or cumoids, exceeds that of the lower, the loading may be relieved from the open spaces and the upper cumoids may restrain the lateral, and therefore the vertical, movement of the lower cumoids, thus adding to their resistive strength.

In an interesting paper read before the winter meeting of the West Virginia Coal Mining Institute, the late F. C. Keighley called attention to the fact that when the lower roof broke or was preparing to break in the mined spaces of the Connellsville region it frequently weakened the lower roof in the rooms and headings nearby, despite the strong support afforded by large pillars. This caused in the narrow places many falls which had to be loaded out.

It would seem, therefore, that the breaking of the lower roof or its initial stressing tears the lower roof from the upper and from the ribs, converting it into a cumoid structure which is too weak to stand the strains to which it is exposed. In cases, however, these primary failures may be due to substitution of a pyoid for a holoid structure. For it must not be forgotten that where the bottom, ribs, and roof are one and indivisible, the floor is an element of strength and prevents the roof from breaking. Just as the lower flange in a rail helps the ball of the rail to support the weight, so does the floor help sustain the roof so long as the former is unbroken.

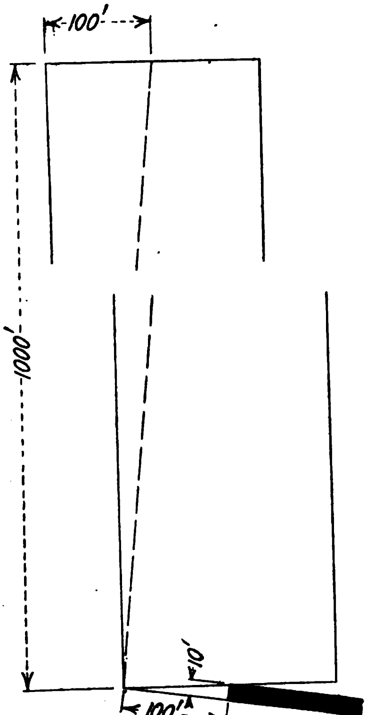

FIG.17

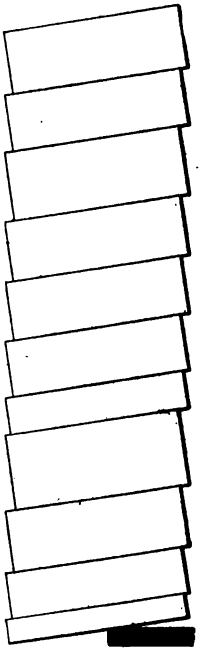

FIG.18

Fig. 17 shows the large amount of distortion needed to permit the fall of one leaf of a broken cumoid, or conchoid, 1,000 ft. deep.

Fig. 18 shows how this distortion is secured by repeated horizontal shears at places of weakness.

One of the patented features of the Luten bridge construction is its holoid structure. Luten does not use the term, but nevertheless the construction is closely analogous to the structure I have been describing. Luten builds not one bridge but two, one for vehicles to pass on and one to bind the feet of the piers together so as to form a perfect holoid. Destroy the lower bridge and the upper or traveling bridge is in a precarious condition. (See Fig. 15.)

The holoid structure gives the strongest of roofs; the pyoid is less strong; the cumoid is still weaker. Every horizontal shear makes the roof progressively less stable. Strange to say, vertical cracks permit the replacement in many cases of a less stable by a more stable structure. The fractures in a vertical direction, which would ordinarily be thought more detrimental than those in any other direction, prove not unimportant, it is true, yet in a way strengthening.

When a cumoid of great depth finds itself inadequate to its own support as a cumoid and begins to rend vertically from the surface downward to its supports and upward from the span centers to the surface, then the cumoid beam becomes an arch, and the cumoid plate becomes a dome and until it fails in its new structural relations, it cannot continue to extend those rents which threatened its stability. Let us say that the cumoid has become a conchoid (from *conchus*, a shell), the lines of stress forming figures resembling the shell of a bivalve.

As the elements of which the roof consists crowd each other into the opening it is impossible for the roof to rupture or fall until the various strata cease to act as a unit and begin to slide past each other. (See Fig. 16.) If we imagine an element of rock stretching from a rib to the half span, a distance, let us say, of 100 ft.; if we further suppose the excavated coal or other mineral to be 10 ft. thick, and regard the rib, floor, and falling rock element to be so adamant that none of the three will crush, heave or bend, then the inclination of the element when one corner reaches to the floor will be 10 per cent.

If the excavation is 1,000 ft. below the surface, the displacement of the block at the surface would have to be 100 ft. (See Fig. 17.) At a greater depth the displacement would be more. It is needless to say such a displacement cannot occur even where the crop, broken roof, or the sags in nearby workings make a certain amount of lateral adjustment possible. But horizontal shear along weak planes, such as beds of clay or coal or stratification planes, will permit such a number of readjustments in the element of rock itself that it will be able to come down. It will, in fact, rather *tend* to tilt than actually upset forward. (See Fig. 18.)

This, then, is how the end comes unless the rock is too strong to shear horizontally. Subsidence ceases when the required deformation has taken place. The choking up of the falls probably has but little to do with the final result, though it may have its effect in some cases. In fact, the theory as given is comforting to the conservationist, as the beds above the one extracted suffer but little, being exposed only to the horizontal shearing process, which is by no means destructive of the integrity of the measures.

However, the rupture at the half span may often result in falls of moderate size at that point, for when the monolithic roof is weakening by successive horizontal shears, we have a series of superposed cantilever cumoids entirely unequal to the task of self support. If the action which forms these cumoids does not let them all down to the floor suddenly, they will be sure to fracture successively from the bottom upward, and such seams of mineral as partake of this independent action will be destroyed.

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Kick vs. Rittinger: An Experimental Investigation in Rock Crushing, Performed at Purdue University

BY ARTHUR O. GATES,* B.S., M.E., MISHAWAKA, IND.

(San Francisco Meeting, September, 1915)

INTRODUCTION

RITTINGER'S law of the energy expended in crushing is, as roughly stated by Professor Richards,[1] that the work of crushing is proportional to the reduction in diameter; or, as I have more fully expressed it:

> "The work done in crushing is proportional to the surface exposed by the operation; or, better expressed for this purpose, the work done on a given mass of rock is proportional to the reciprocal of the diameter of the final product—assuming that all the mass has been reduced to one exact size, which is only theoretically possible."[2]

Kick's law is stated by H. Stadler as follows:[3]

> "The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of these bodies."

In other words, the energy expended is porportional to the volume reduction, instead of the diameter reduction.

That these two laws would give widely different results may be shown by a simple imaginary case. A ton of 16-in. cubes is broken to 1-in. cubes at the first operation, and these are broken to 1/16-in. cubes at a second operation. Since the first operation produces only one-sixteenth as much new surface as the second, the ratio of energy expended in the two operations would be by Rittinger's law as 1 to 16. By Kick's law, since the volume-reduction ratio is the same in both cases, the ratio of energy would be 1 to 1. This discrepancy is great enough to challenge a test by actual experiment, of which there has been, hitherto, but little compared with the amount of argument. The present paper is the record of such a test, and is offered as showing that Rittinger's theory more nearly represents the actual facts that any other proposed hitherto.

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* Sales Engineer, Dodge Sales & Engineering Co.

[1] *Ore Dressing*, p. 304 (1903).

[2] *Engineering and Mining Journal*, vol. xcv, No. 21, p. 1039 (May 24, 1913).

[3] Grading Analyses and Their Application, *Transactions of the Institution of Mining and Metallurgy*, vol. xix, p. 478 (1910–11). The original statement will be found in *Das Gesetz der proportionalen Widerstände*, by F. Kick.

Mr. Stadler, in the paper already cited, applying to rock fracture Kick's law (which I accept as applicable to the deformation of elastic bodies), proves that the stamp mill, working on relatively coarse feed, is several times as efficient as the tube mill on fine feed, this relation being expressed in what he calls "relative mechanical energy per horsepower." He distributes his energy units among the various sizes produced, according to Kick's law. But my results show, among other things, that many more of Stadler's energy units are obtained per foot-pound-applied in coarse than in fine crushing. This suggests a doubt as to the correctness of his unit, since one would expect a foot-pound-applied to produce the same number of "energy units," whether the feed and resulting product were coarse or fine. When Rittinger's theory is applied, the number of mesh-tons (my unit of surface produced) is nearly proportional to the foot-pounds-applied, whether the product be coarse or fine; hence my conclusions favor the simple Rittinger theory that surface produced is proportional to energy applied.

The Two Laws

1. It is the purpose of this paper to record and interpret a series of experiments made for the purpose of studying the consumption of energy in the crushing or fracturing of rock particles both coarse and fine. At the start, the expectation was to determine constants according to the Rittinger theory for several rocks, whereby it would be possible to calculate absolute crushing efficiencies of machines crushing such rocks, or to predetermine what a machine would accomplish in the way of tonnage and screen analysis upon a given rock, knowing what the machine accomplished upon another rock whose constants were likewise known. While approximate constants for a few rocks were determined, the results indicated that an amount of work (perhaps with the microscope upon sizes too small to be screened) greater than could be accomplished by a single investigator would be necessary to determine these constants with reasonable accuracy. For this reason no constants are submitted as such in this paper.

2. It is possible, however, to compare by means of these experiments the two antagonistic "laws" of crushing already mentioned. The fracture of cubes by compression in a testing machine produces, in nearly all cases, pyramids of the same general shape (Fig. 1), indicating that the stresses are greatest along certain planes of fracture; and, since the relative movement must have been greatest along these fracture or shear planes, the greatest amount of energy must have been expended there. While I know of no tests that show that the several pieces return to the

dimensions existing before the break, it seems reasonable that they do, and that they do not retain the deformation. All that has resulted from the pressure and consequent movement is *cracks*. They are new, while the particles resulting are of the same form and specific gravity as they were before separation, and if put together accurately the dimensions of the mass would be the same.

Rittinger's Law

3. Rittinger's theory is the law of the *cracks*. It is well explained in an article by E. A. Hersam in the *Mining and Scientific Press*, an abstract of which is given in the third volume of Richard's *Ore Dressing*, and to which I acknowledge my indebtedness for some of my ideas and methods.

4. The mathematics of this theory follow:

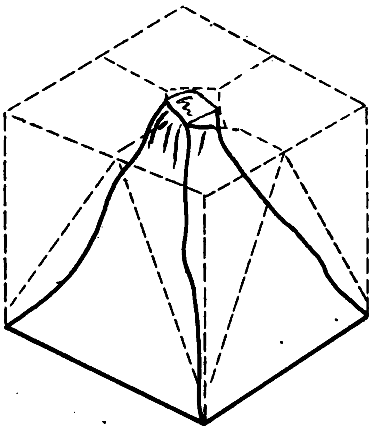

FIG. 1.—FRACTURE OF CUBE IN PRACTICE.

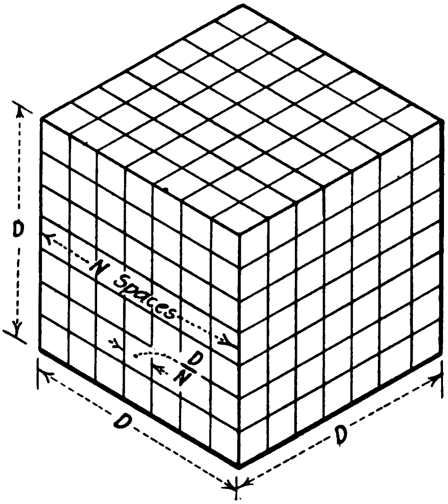

FIG. 2.—THEORETICAL FRACTURE.

Assume a cube of rock (Fig. 2) of side D, divided into smaller cubes by planes passing through it in three directions, as shown. The surface of the original cube being $6D^2$, the aggregate surface of the smaller cubes will be $6ND^2$, and the new surface exposed will be $6(N - 1)D^2$. The theoretical mesh or reciprocal of diameter of new particles, M, is $\frac{N}{D}$, whence $N = MD$; and the aggregate final surface, $S = 6ND^2 = 6MD^3$.

With D constant, that is, starting from the same size of cubes in both cases, $S : S' :: M : M'$, or the total surfaces of equal volumes of rock, each composed of particles of uniform size and shape, are to each other as the theoretical mesh or the reciprocal of the diameter of the particles composing the respective masses.

Suppose the surface S to be made up of x_1 parts (by weight) of M_1

mesh (theoretical), x_2 of M_2, x_3 of M_3, etc. . . . . . x_n of M_n. Then,

$$S = k\Sigma_1^n xM;$$

that is, the summation of the weights by the theoretical meshes, where k is a constant, six times the volume of unit weight of the rock. If different rocks were considered, specific gravity would enter into the value of k.

Likewise,

$$S' = k\Sigma_1^n x'M'$$

Now where S was the surface existing on the mass before breaking, and where S′ is the total surface existing after the breaking, the new surface

$$S' - S = k\Sigma_1^n(x'M' - xM)$$

For a given screen analysis, weights of the different theoretical meshes, we may plot on co-ordinate paper x against M and x′ against M′, and obtain a crushing-surface diagram (Fig. 3) representing the equation above given for new surface, the area between the steps being proportional to the difference in surface area. If we take closer divisions of M, smaller differences in x will result, and the limits of such divisions will result in the smooth curves, the area between which is a true measure of the surface produced.

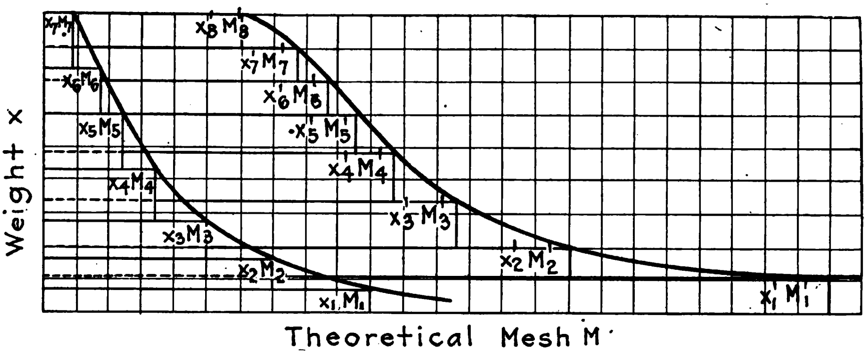

FIG. 3.—CRUSHING-SURFACE DIAGRAM.

Thus, on the Rittinger theory that surface produced is proportional to energy applied, the area of the diagram is a measure of the energy going into the crushing. (It should be distinctly understood that bearing- and gearing-friction losses should not be included in the energy-applied measured by the crushing-surface diagram, and that it is therefore impossible to compare the two laws by tests upon commercial machines, unless the friction losses are known.)

THE CRUSHING-SURFACE DIAGRAM

The crushing-surface diagram has the advantage over any other graphic method of comparing screen analyses, that it shows at a glance the "classification" of the pulp, the distribution of the surface, and (according to the Rittinger theory) the energy spent in its production. As the cards of a steam engine enable us to determine the power developed, and to locate faulty operation or design, so the crushing-surface diagram enables us to determine the power consumed, and to locate points at which the crushing practice may be improved and faults remedied in design.[4]

5. From my original article on The Crushing-Surface Diagram,[5] I take Fig. 4 and the following theoretical proof of the Rittinger theory:

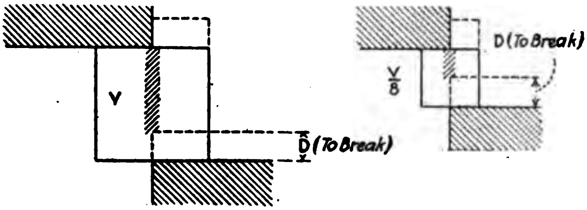

FIG. 4.—RITTINGER'S LAW.

100-Mesh Cube	200-Mesh Cube
Area one section $= A$	Area one section $= \frac{A}{4}$
Average resistance to shear per sq. in. $= F$	
Energy $= FAD$	Energy $= F\frac{A}{4}D$
Surface produced $= 2A$	Surface produced $= 2\frac{A}{4}$

$\therefore$ Energy Proportional to Surface.

Suppose it were possible to hold two similar cubes between the two offset faces as shown, and that forces were applied until the deformation shown by the dotted lines was obtained, it will be seen that only the molecules along the vertical center line [it should read plane] are stressed and deformed, the mass of the cube away from this surface receiving practically no pressure or deformation. The energy in this case required to produce rupture will be the product of the average resistance to shearing per square inch by the area along which rupture takes place, and by the distance the two offset faces move together. The average resistance to shearing is a variable quantity, as the deformation increases up to rupture. To reduce to cubes, this amount of energy must be multiplied by three, as three similar fracture planes must be made to produce cubes. And it will not be hard to see that the distance through which the offset faces must move in either case must be the same and not proportional to the thickness of the piece. To break the molecular bond between adjacent particles would require the same movement, regardless of the thickness of the piece.

"It will be noted that if eight of the half-diameter cubes be sheared to produce cubes of half their size, the new surface presented will be double that formed when the single large cube is sheared into half-size cubes, and also that the energy required in the case of the smaller cubes is double that required in the case of the larger cubes. This should demonstrate that energy applied to crushing is proportional to the surface produced."

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[4] This matter has been discussed in my article, Applications of the Crushing-Surface Diagram, *Engineering and Mining Journal,* Apr. 18, 1914.

[5] *Eng. and Min. Jour.,* May 24, 1913.

Of course the above passage states a theoretical case of shear, and it is not to be expected that cubes can be held in the manner shown; in practice the breaks occur, perhaps, as in Figs. 1 and 5.

6. New units are apparently necessary when using the crushing-surface diagram. The term "mesh-grams" as a proposed unit for crushing constants and measurement of surface I have defined as "the surface produced by a theoretical crushing operation in which one gram of particles of the same diameter are all reduced to a diameter whose reciprocal is one greater than before reduction." The unit "mesh-ton" I have similarly defined as "the increased surface produced by crushing all particles of a ton of rock to a diameter whose reciprocal is one greater than in its previous condition."[6]

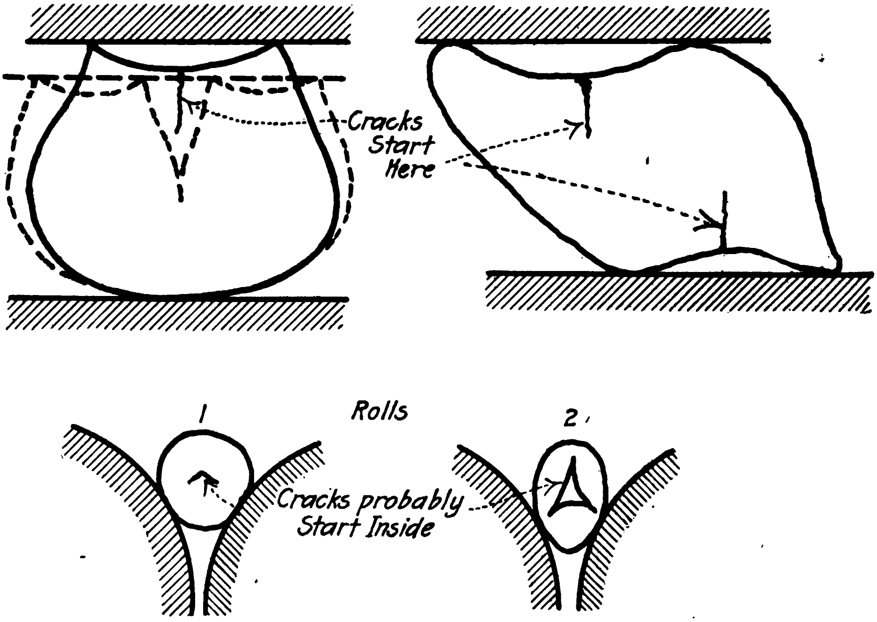

FIG. 5.—BEGINNINGS OF FRACTURE.

As the crushing contemplated in the definitions is only theoretically possible, a couple of examples may explain them better. Shapes remaining the same, preferably cubes for argument's sake, during all stages of the crushing, if 1 g. of 0.1-in. rock (reciprocal, 10) be crushed until it is all 0.05-in. (reciprocal, 20), the new surface produced is 10 mesh-grams. Or if 10 g. be reduced to exactly 0.01-in. from exactly 0.1-in. particles, there will be an increase of 10 times 100 minus 10 times 10, or 900 mesh-grams. Mesh-pounds, mesh-ounces, mesh-percentages, etc., are similarly justified and defined.

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[6] *Trans.*, xlvii, 59 (1913).

7. The term "mesh" for a measurement has been objected to, as ambiguous in meaning. Some other short term may be desirable to take its place in forming these compound names for surface produced. But nothing better has been suggested. "Reciprocal" (of diameter) is too general and too long; "theoretical mesh-tons" is likewise too long and is also liable to misinterpretation; "dia-ton" has been suggested; but it would hardly do to speak of "dia-grams" in connection with a diagram. Perhaps "pitch-grams," "pitch-tons," etc., would be satisfactory, pitch being defined as the theoretical mesh or reciprocal of diameter(probably in inches). I would here again point out that these quantities are measures of surface, and that the weight element is only permitted to enter on the basis of equal specific gravity; where the specific gravity varies, a correction must be made for it.

8. Again I quote from my original paper my interpretation of Kick's law, Stadler's statement of which I have given above:

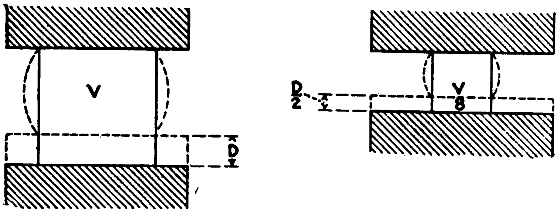

FIG. 6.—KICK'S LAW.

100-Mesh Cube — 200-Mesh Cube

Area one face $= A$ — Area one face $= \frac{A}{4}$

Average resisting force per sq. in. $= F$

Energy $= FAD$ — Energy $= F\frac{A}{4}\frac{D}{2} = \frac{FAD}{8}$

$\therefore$ Energy Proportional to Volume.

"This law does not apply so much to crushing as to deformation of bodies before rupture takes place.

"In Fig. 6 are represented two particles of ore 'of equal technological state,' shown as cubes between the faces of a crushing or testing machine. Assuming the theoretical mesh, equivalent to the reciprocal of diameter, and using concrete values, we have a 100-mesh particle with eight times the volume of the 200-mesh particle, and with an area per face four times that of the 200-mesh particle. The dimensions are as two to one, and the bodies being similarly deformed within the elastic limit without fracture, the energy that must be applied in each case to produce this deformation is the product of the average resisting force per square inch, the same in both cases, by the area worked against and by the distance through which this average force works. As shown in the figure, in this particular instance the energy absorbed is proportional to volume, and it can be similarly shown for the general case. On the gradual release of the external pressure the energy absorbed is given back to the machine producing the deformation [better, which produced the deformation] and the body returns to its original shape. It should be noted that the body has been deformed only by a gradually increasing pressure, the first increment of deformation not requiring so much pressure as the last. *In case the body has been deformed beyond its elastic limit* *either the whole mass of particles have been reduced to the molecular state by the freeing of their bonds with adjacent particles, which never happens, or fracture takes place along a few surfaces by the breaking down of some of the weaker bonds,* and the particles thus formed are free to resume their original shape in so far as they are not held beween the machine surfaces. The energy given up by them is probably used in some sort of lever action in making fracture planes. So the energy absorbed according to Kick's law does not stay in the particles after pressure is released and therefore this does not govern to any great extent the amount of energy absorbed in crushing."

Kick's Law

It is evident to the engineer that Kick's law as explained above is but another form of the law of elastic bodies: namely, that within the "elastic limit" stress is proportional to deformation.

10. There is no incompatibility between the Rittinger theory and the laws of elastic bodies such as glass, steel, rock, etc. Up to the elastic limit, deformation is proportional to stress, and absorbed energy is proportional to the volume of the particle stressed, according to Kick's law. In Fig. 7 the area Oyd is a measure of this energy. When pressure within the elastic limit is released the body returns to its former shape, there having been no interior breaks so far. When, however, the elastic limit has been exceeded, certain local ruptures have taken place within the body at points weaker than others; these breaks occur consecutively or simultaneously at various places within the body, and upon release of pressure the body is found upon careful measurement to have taken on a permanent "set." The first fracture noticeable occurs after several ruptures have weakened the body at one point, which may be along a plane of greatest resultant stress, or of structural weakness. Referring back to Fig. 7, the elastic limit having been exceeded, as pressure is gradually released, its relation to the deformation is shown by the line af, the area afn being a measure of the energy given back to the machine

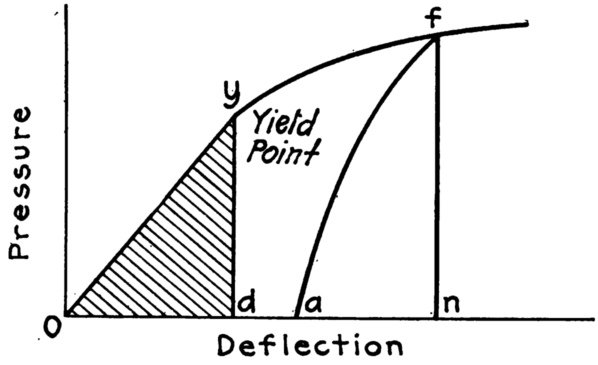

FIG. 7.—STRESS-STRAIN DIAGRAM.

(by Kick's law), while the area Ofa is a measure of the energy absorbed by the local breaks, which may or may not have become visible. In the case of a wrought-iron or mild-steel test bar undergoing tension between the jaws of a testing machine, local ruptures are probably occurring throughout the bar, reducing the section at those points of weakness, and putting greater load upon adjacent particles which may be then able to carry the load better.

The final rupture of the steel bar occurs when a number of local breaks at one particular section have so weakened the bar that all the work the machine is doing is concentrated at that section, as shown by the "necking in" just before failure takes place. Unquestionably in spite of the permanent set in the test specimen away from the point of failure, the majority of the crystals composing the specimen are in the same form as before the test, but some have slipped away from others and there is a new adjustment due to the failure of the few which could not hold on, the others at that section being in position to support the load for a further period. If it were wrought iron, we could say the fiber slipped along the slag or impurities between fibers until the slipping fibers were crowded so that they obtained a sort of "strangle hold."

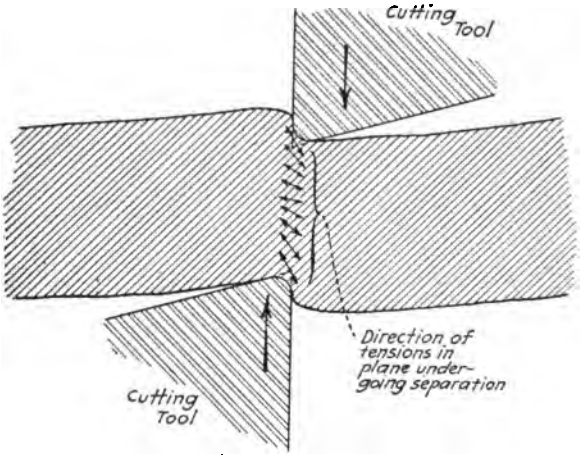

FIG. 8.—SHEARING METAL BAR IN COMMERCIAL MACHINE.

10. In the case of the shearing of iron and steel bars, it has been shown[7] that the depth of penetration (the distance traversed by the cutter before separation takes place) is not proportional to the thickness of the metal, as it would be in case Kick's law applied, but is proportionally greater in thin sheets than thick—about 75 per cent. of the thickness in the case of sheets $\frac{1}{8}$ in. thick, and about 25 per cent. in the case of sheets 1 in. thick.

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[7] *American Machinist*, 1905; see references in J. D. Hoffman's Machine Design.

On the face of it, this fact does not uphold the theoretical argument for Rittinger any more than for Kick, but let us look farther.

In a commercial shearing machine we do not get true shear, but a condition shown in Fig. 8. The material is compressed under the blades of the shear, which are themselves somewhat distorted during the operation, and failure is due to tension in a line inclined to the plane of the cutting edges. In the case of a thick plate there would be more deformation of the cutting edges due to the higher resistance of the material, and more of a width concerned than in the case of thin sheets. When a hole is punched in steel, there is a volume of material around it that must be removed for first-class work; this volume is probably full of minute fractures the area of which cannot be measured. I am told that rock crushed for concrete, is seen upon microscopic examination to be full of minute fractures, due to the crushing, which cause each piece to be weaker than a natural pebble of the same mineral and size. If the rock were uniformly full of these invisible fractures, Kick's law would apply in spite of all our theoretical reasoning. Just as in the case of the hole punched in steel, I believe that these minute fractures in the rock are close to the fracture planes.

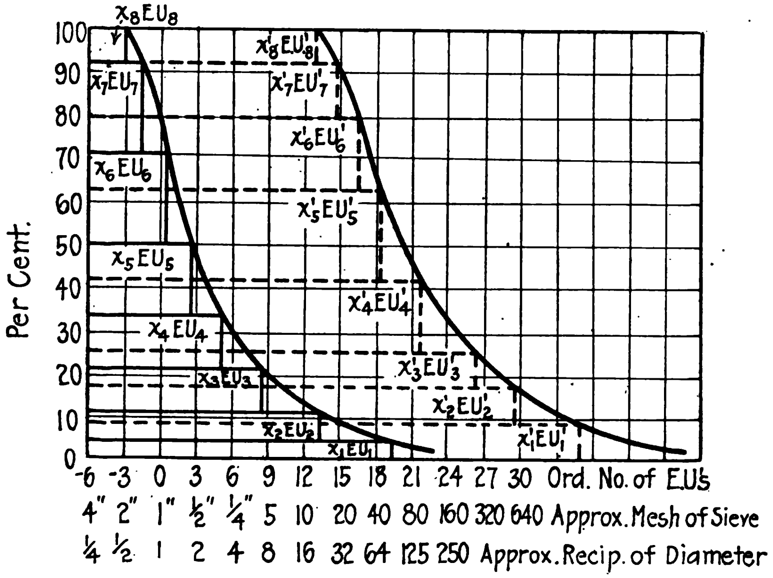

FIG. 9.—MECHANICAL VALUE DIAGRAM.

11. A. F. Taggart, has discussed[8] Kick's law thoroughly from the standpoint of its advocates and there is no reason for a repetition of his work. However, taking it at their valuation, and using data obtained from Stadler's papers already referred to, we can by reasoning similar to

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[8] The Work of Crushing, *Trans.* xlviii, 153 (1914).

that already employed develop a diagram for energy absorbed similar to the crushing-surface diagram. (Fig. 9.)

With constant weight and similar shapes, Stadler allows one energy unit (E. U.) for each reduction to one-half the volume immediately preceding. Expressed mathematically,

$$\left(\frac{D}{d}\right)^3 = \frac{V}{v} = 2(\text{E. U.})$$

Or, as more frequently expressed,

$$\log V - \log v = (\text{E. U.}) \log 2$$

Whence

$$(\text{E. U.}) = \frac{(\log V - \log v)}{\log 2} = 3\,\frac{(\log D - \log d)}{\log 2}$$

D and d being initial and final dimensions, V and v being initial and final volumes. For example: Reducing 1-in. cubes to one-half volume, 1 E. U. has been expended; reducing to one-fourth volume, 2 E. U.; to one-eighth volume, 3 E. U.; to one-fourth dimensions, 6 E. U.; to eighth dimensions, 9 E. U.; and so on. Reducing from 100 in. diameter to 1 in. the same number of E. U. are expended as from 1 in. to 0.01 in., or as from 0.01 in. to 0.0001 in. (all on quantities of the same volume, both here and in the previous sentence). Of course various sizes are produced in crushing, and each grade or size has a different value, the same reasoning applying as in section 4, above.

Suppose mechanical value W be made up of x_1 per cent. of E. $U._1$, x_2 per cent. of E. $U._2$, etc. . . . . . x_n per cent. of E. $U._n$; then

$$W = c\sum_1^n x\ \text{E. U., before the crushing operation, and}$$

$$W' = c\sum_1^n x'\ \text{E. U}'.\ \text{after crushing.}$$

The new mechanical value, $W - W' = c\sum_1^n (x\,\text{E. U.} - x'\,\text{E. U}'.)$.

Plotting x against E. U., we have, when the limiting curves are drawn, a diagram (see Fig. 10) similar to the crushing-surface diagram, the area of which is a measure of the mechanical value of the crushing operation or the energy units (E. U.) absorbed. (Percentages should be expressed decimally in the above calculations.) The mechanical value of sands given in E. U. from the diagram is multiplied by the tonnage per day (or hour) and divided by the horsepower, giving the *relative mechanical efficiency.*

12. The E. U. per foot-pound applied upon 100 g. as found from these mechanical-value diagrams compared with mesh-grams per foot-pound as found from crushing-surface diagrams for the same crushing operation will, if covering a range of operations show which of the two laws under consideration is consistent with the facts. This is the method used in

comparing my results. (In Fig. 9, it will be noted that the theoretical mesh or reciprocals of diameter are designated at the corresponding "Ordinal numbers of energy units," which is self-explanatory, and perhaps permits easier comparison with the crushing-surface diagram for the same crushing operation.) The following comparisons based upon the experimental work performed show that the Rittinger theory applies to crushing operations and that Kick's does not. There are discrepancies when we apply the Rittinger theory; but these can be explained reasonably by considering the expenditure of energy upon the sizes finer than our screen permits our measuring, which must present considerable surface, if we assume that the curve of the screen analysis continues to follow the same law into the finer sizes as among those already measured.

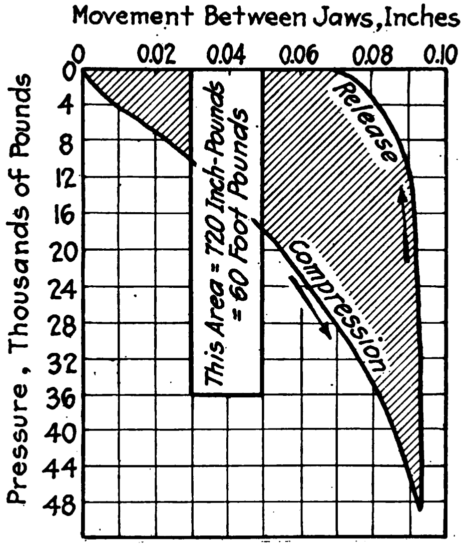

FIG. 10.—ENERGY DIAGRAM.

METHOD OF TESTS

13. In this work one or many pieces of rock were compressed between the faces of the Amsler-Laffon hydraulic testing machine in the Laboratory for Testing Materials of Purdue University. (This machine is described in the appendix.) The movement between the faces (similar to those of a crusher) was measured by a deflectometer having a vernier on which movements of 0.001 in. could be read, while the pressures between faces was read upon a dial to a maximum of 150,000 lb. The product of the average pressure (or rock resistance) by the distance through which that pressure was exerted is work or energy, and easily reduced to foot-pounds.

I found it easier to plot my results as obtained, pressure against distance, as shown in Fig 10, the area of the closed curve, completed by the record of pressure and distance when pressure was gradually released, being a measure of the energy in inch-pounds (divided by 12 equals foot-pounds). This energy as foot-pounds, or as horsepower-hours, can be compared with the surface produced as shown by the crushing-surface diagram of the screen analyses of initial and final products of the operation; and it can likewise be compared with the energy units (E. U.) of Stadler, as shown by mechanical-value diagrams similar to Fig. 9. By crushing the same amount of rock from coarse to medium, from medium to fine, from fine to very fine, etc. (coarse and fine being merely relative terms), the two laws may be compared through a sufficient range to detect any abnormal results or inconsistencies.

14. To each test, whether it involved one or several consecutive crushing operations, was given a letter by which it is designated throughout this paper. My records are in the form of crushing-surface diagrams with the weights marked to place as plotted. From these the mechanical-value diagrams must necessarily be derived, as this work was not undertaken primarily to settle the case of Kick vs. Rittinger. The pressure-deformation curves were plotted directly in a similar manner without recording numerical readings. The advantage of recording by plotting the observations directly upon co-ordinate paper is the speed at which it is done, and the check afforded by the shape of the curve. It is hoped that the work will be acceptable in its present form, as numerical tabulations would add nothing of value to this paper. So far as possible the results of calculations are given in the diagrams, and hence it will be unnecessary for the reader to refer back and forth from text to diagram to any great extent.

15. As the crushing-surface diagram does not depend upon any definite system of screens so long as the aperture be known, all the screens used were measured with a Bausch & Lomb microscope having a scale reading to $\frac{1}{500}$ in., and the reciprocals of diameter of these apertures were used in plotting.[9]

Certain of the finer sizes were screened through bolting cloth held in embroidery frames. I have suggested specially prepared plotting paper, *i.e.*, with vertical lines drawn in at the reciprocals of diameter corresponding to the apertures of the screens used, for such cases as the Tyler standard screen-scale sieves, or the I. M. M. standard. Since this paper was first submitted to the Institute and of course since the work was done, I have discovered that cumulative direct diagram sheets with Tyler S. S. S.

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[9] In some of the early work actual screen meshes were recorded, but for the sake to uniformity these have been reduced to reciprocals and shown as though recorded that way.

sieves are desirable for this work and that there is no need of any other sheets for crushing-surface diagrams. Fig. 11 shows a crushing-surface diagram on this paper, the plotting running in the opposite direction from that for which it was designed; 1 sq. in. on the regular sheet, with screen sizes located as indicated, representing 8.85 theoretical mesh-tons of surface per ton of ore crushed.

16. All results have been modified to a basis of 100 g., so that percentages required for use with E.U. are readily obtained, about one-third of the tests having been made upon a definite 100 g. In none of the tests was any attempt made to account for the losses occasioned by fine dust floating away, or by coarser pieces shooting off as they were fractured. In no case could these be more than 2 per cent., which would not materially affect the results. Extreme accuracy was not particularly necessary in this work. It was rather a reconnaissance; and if more than three significant figures are given at any point, too much importance should not be attached to the last figure. At the same time the work was carefully done and the record carefully made, but there is so much unknown that I would not care to defend the fine details of my results.

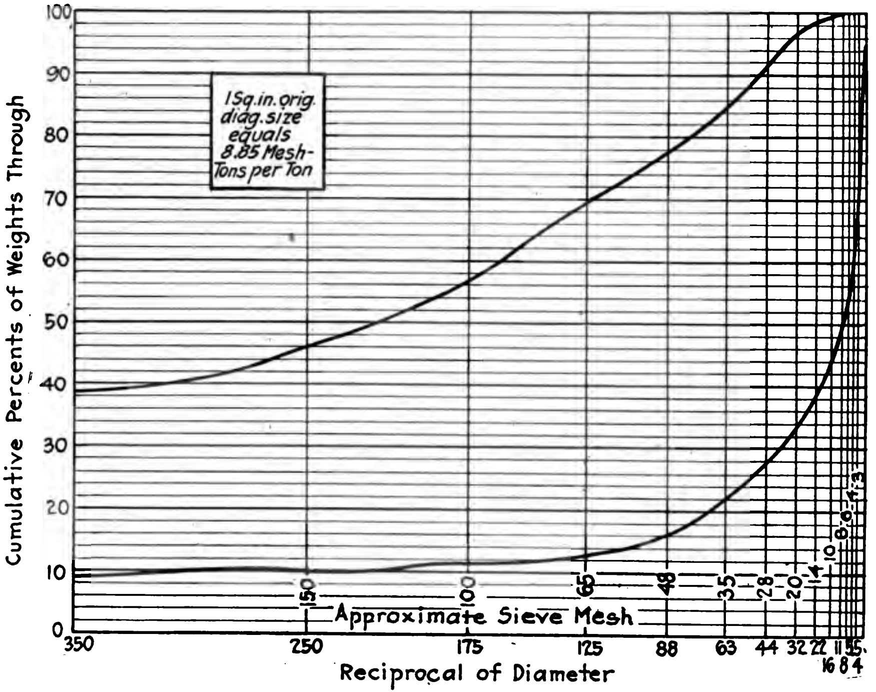


FIG. 11.—CRUSHING-SURFACE DIAGRAM ON TYLER CUMULATIVE DIRECT SHEET.

Record of Tests

17. In Fig. 12, A and B are results obtained upon a tube-mill feed, a fairly siliceous gold and silver ore from Mexico, of which 100 g. of $-\frac{5}{16}$-in. +4-mesh pieces were crushed by two movements of the machine, without

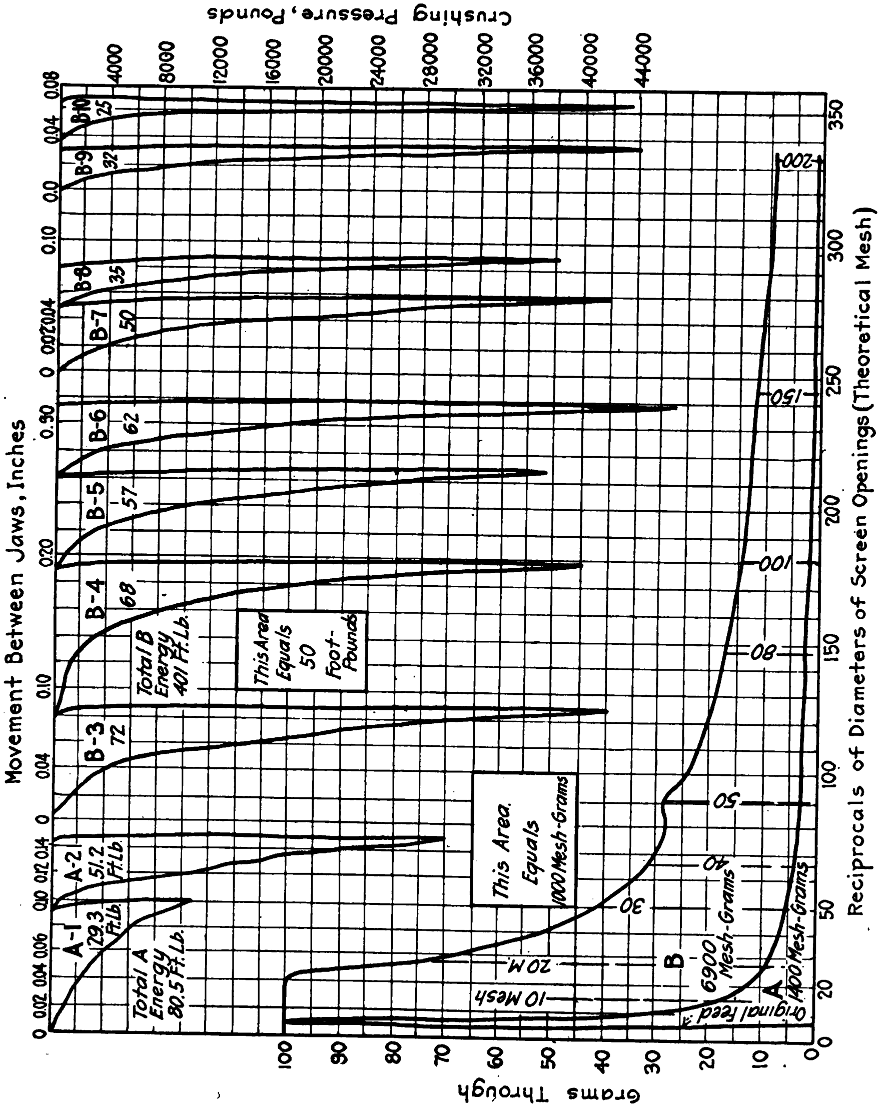


Fig. 12.—Crushing-Surface and Energy Diagrams. Siliceous Mexican Ore.

intermediate screening, the purpose being to crack every piece. The energy is represented by the areas A-1 and A-2. It is of interest to note how the release of maximum pressure is usually accompanied by some movement of the jaw, as in this case, indicating a return of energy to the machine, shown in the diagram by the slight curve back to the zero-pressure line. After the particles adjusted themselves upon release of the pressure it took more movement between the crushing faces to get up to the previous pressure, as indicated by the gap between A–1 and A–2. After sizing and plotting the screen analysis in the form of a crushing-surface diagram A, the particles were re-crushed eight times, each time being screened through a screen of about 30 mesh to remove the fines and eliminate the condition of "choke crushing" as far as possible, the oversize only going back to the machine. The energy areas for these eight crushings are B–3, B–4, B–5, B–6, B–7, B–8, B–9, and B–10; the screen analysis is plotted as a new side of the crushing-surface diagram, B. The areas of the energy diagrams A–1 and A–2 were found by planimeter to be 1.40 and 2.38 sq. in. respectively, a total of 3.78 sq. in. One square inch, for the scales used, being equal to 250 in.-lb. or 20.83 ft.-lb., 3.78 times 20.83 or 80.5 ft.-lb. of energy was expended in making the crushing A. By similar reasoning, the area of the crushing-surface diagram A, as found by planimeter (as far as screening was carried out, to the 200-mesh sieve), multiplied by the value of its unit area in mesh-grams, shows that 1,400 mesh-grams of surface was produced. From these two quantities, a value of $\frac{1400}{80.5} = 17.3$ *mesh-grams per foot-pound* (+200-mesh sieve) was obtained, as the constant to be used in calculating efficiencies, etc., for this rock. Note, however, that the mesh-grams entering into this constant are only those "plus" 200-mesh sieve, and that there are perhaps more mesh-grams in the "minus" 200-mesh than in the "plus," and they must at some time be taken care of in calculations. It is these "minus" mesh-grams which will account for the discrepancies in results by Rittinger's theory, and which will not account for discrepancies in results by Kick's law.

18. For B, the total energy expended was 401 ft.-lb., producing 6,900 mesh-grams (+200-mesh sieve) giving a unit value of 17.21 *mesh-grams per foot-pound* "plus." Fines were screened out after each jaw movement, to reduce "choke crushing."

19. Tests E, F, G, H, J, K, L, M, and N were made upon Bedford limestone, the first in July, 1912, and the rest in July and August, 1913. This is one of the best known building stones in the United States. It occurs in great beds in southern Indiana, is oölitic in texture (weathering brings out myriads of fossils) and, because of its apparent uniformity, is probably as good a rock as could be selected for a standard. (I reserve the right to change this opinion later.) The particular specimens came from stone furnished for the Purdue Library building.

20. The crushing-surface diagrams of E and F (Figs. 5 and 4 in my original article in the *Engineering and Mining Journal* of May 24, 1913) are

not shown here. In E, practically all the material was broken in the machine, whereas in F, a great many pieces between 30 and 80 mesh were not broken, but simply had their edges rubbed off. My records as to F

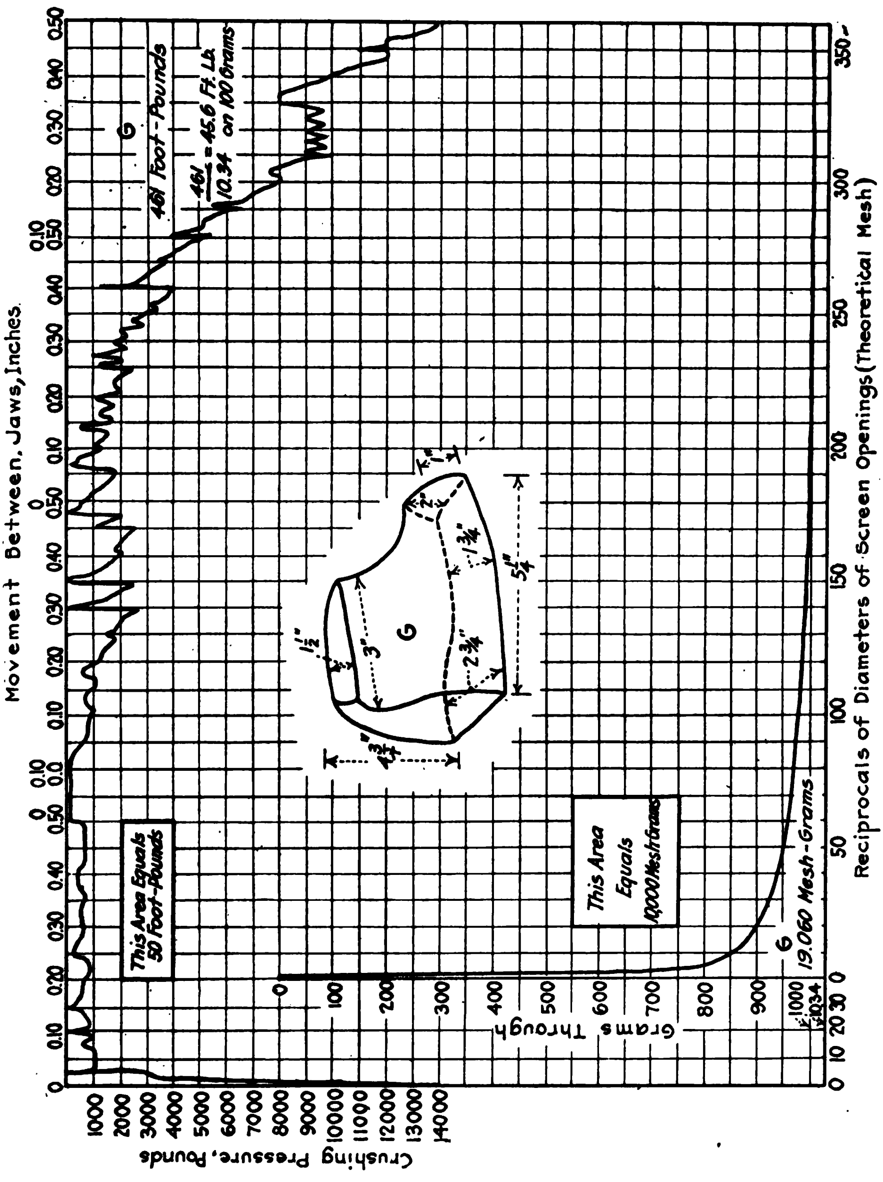


Fig. 13.—Crushing-Surface and Energy Diagrams, and Feed-Sample. Bedford Limestone.

do not clearly show whether I screened betwen crushings; this was done in the case of E.

21. Test G was on a block of the diemensions shown in the small

sketch in Fig. 13, having a superficial area of 63 sq. in. and a weight of approximately 1,031 g. The energy curve is rather long drawn out after the first break, and the irregularities in this curve may be explained

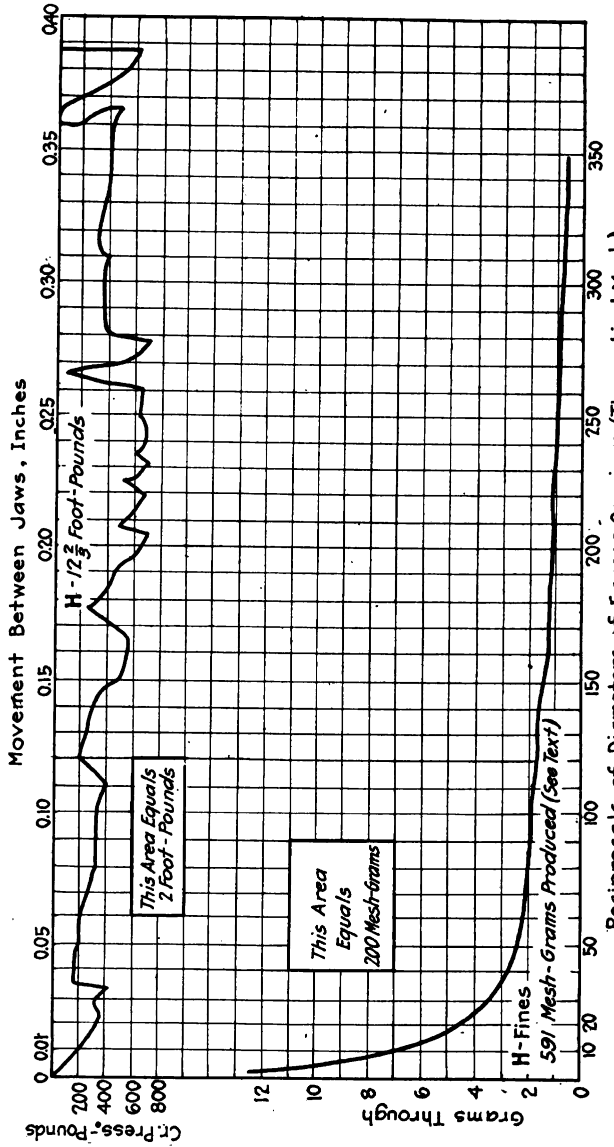


FIG. 14.—CRUSHING-SURFACE AND ENERGY DIAGRAMS. BEDFORD LIMESTONE.

as due to frequent ruptures of fairly large pieces where the load drops back almost to zero. A total movement of nearly 2 in. is recorded: here only one or two pieces perhaps were resisting, while in some of the other cases a great many pieces were resisting at one time. At the end of the first ½ in. of movement the finer pieces were screened out, and the larger returned to the crushing zone; these pieces being rather flat; about 2 per cent. of −200 mesh being produced.

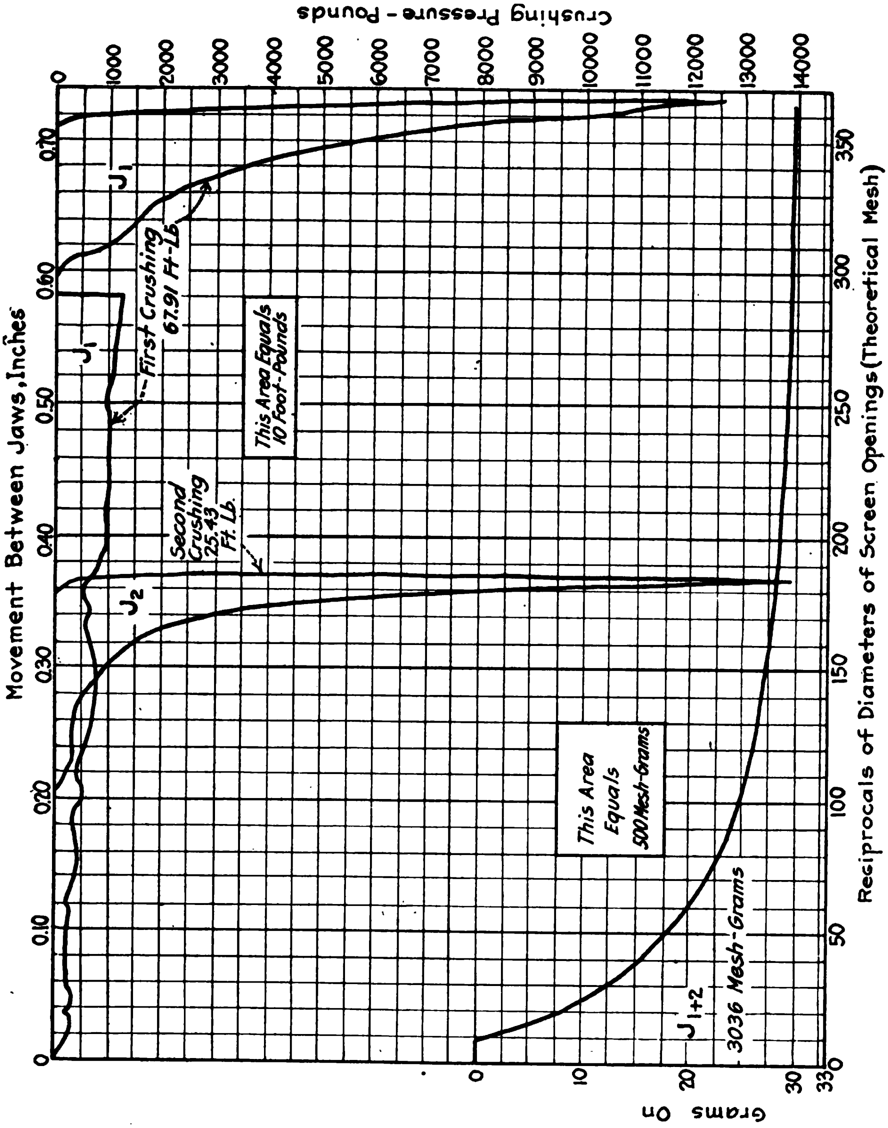


FIG. 15.—CRUSHING-SURFACE AND ENERGY DIAGRAMS. BEDFORD LIMESTONE.

22. H, shown in Fig. 14, covers the results of crushing four pieces having a total superficial area of 30.25 sq. in. and a total weight of 175.5 g. After crushing, eight pieces weighing 163 g. had an area of 30.25 sq. in., just offsetting the original area, while the remaining 12.5 g.

were screened, and the results were plotted as a crushing-surface diagram. 12.66 ft-lb. were expended in crushing the 175.5 g., equivalent to 7.2

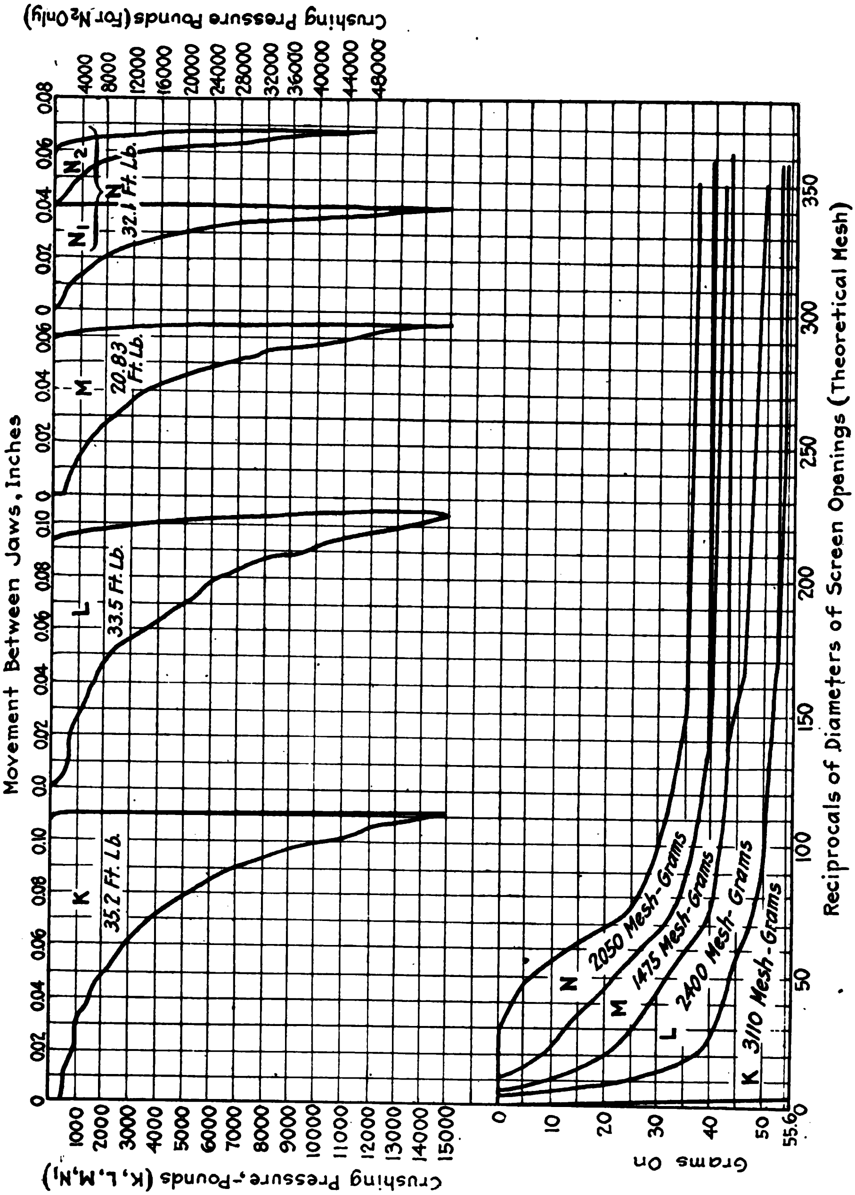
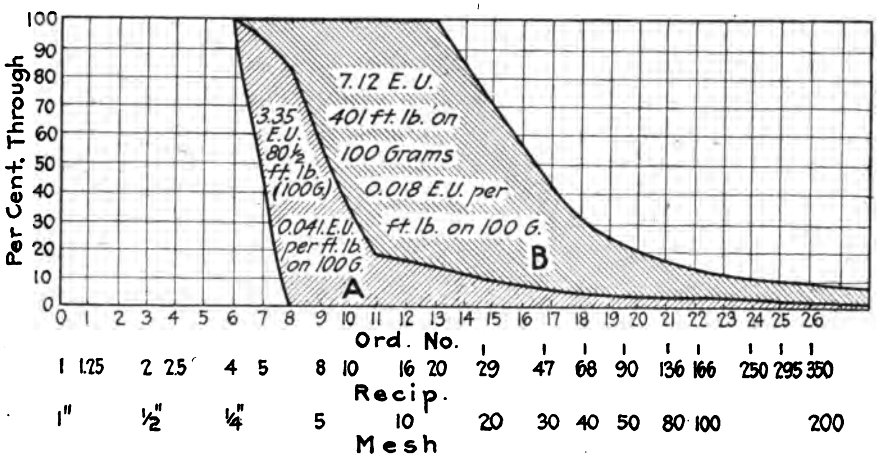


FIG. 16.—CRUSHING-SURFACE AND ENERGY DIAGRAMS. BEDFORD LIMESTONES.

ft-lb. per 100 g.; the eight pieces noted above were estimated to be the equivalent of 60 per cent. of 1 in., 32.5 of ½ in., and 7.5 of fines, as recorded in the crushing-surface diagram. The energy curve is irregular as in the case of G and for the same reason, movement was 0.4 in.; there was no intermediate screening.

23. In J (Fig. 15) a single rectangular piece about 1 by 1.25 by 1.25 in., weighing 33 g., was crushed to pass a 10-mesh sieve, with one intermediate screening. After a deformation of about 0.6 in. was recorded without great pressure, the pressure goes up very rapidly, probably by reason of the packed condition of the particles, a great number resisting crushing. After screening out the −10 mesh, a similar energy curve is obtained on the +10-mesh pieces. The crushing-surface diagram represents the combined product from both crushings. In both cases, the packed condition, in which the particles rub on one another and produce much −200 mesh, accounts for the low value of the mesh-grams per foot-pound.



[Fig. 17.—Mechanical Value Diagram. Siliceous Mexican Ore.

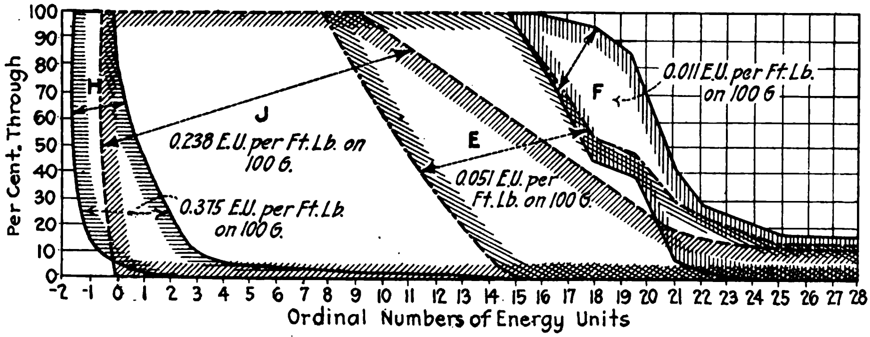


Fig. 18.—Mechanical Value Diagrams. Bedford Limestone.

24. Fourteen very flat pieces weighing 55.6 g., with about 20 sq. in. of surface, averaging in size about 0.75 in. sq. by ⅛ in. thick, were sub-

jected to four separate crushing operations, K, L, M, and N, as shown in Fig. 16. The first crushing K was continued until every piece was found to have been broken at least once; screen analysis was then made and recorded in the crushing-surface diagram, and the material +100 mesh was recrushed as L. M is the result of crushing in a similar manner the +80 mesh from L; and N is similarly the result from the +80 mesh of M. Each crushing was carried as far as the record shows without attempting to remove any fines, although in the case of N pressure was released once and then reapplied, as recorded, in an endeavor to apply more energy. The base lines of L, M, and N separate from the discharge or product lines of the preceding operations, showing the smaller quantities of material crushed in each case.

25. In Fig. 17 the results obtained in A and B on the siliceous Mexican ore have been plotted in a mechanical-value diagram. For the coarser crushing A, 0.041 E. U. per foot-pound-applied on 100 g. was the result, while in the case of B only 0.018 E. U. (etc.) resulted, 125 per cent. *more* being produced when the coarser crushing was done. If graduated crushing had not been done, more energy would have been expended upon the −200 mesh; and I believe this discrepancy would then be greater.

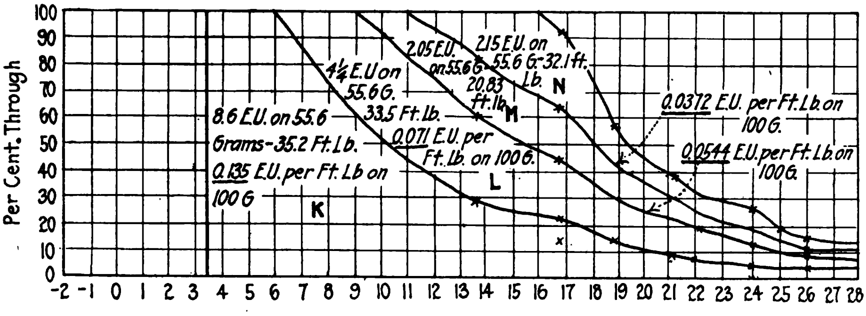


FIG. 19.—MECHANICAL VALUE DIAGRAMS. BEDFORD LIMESTONE.

26. In Fig. 18, E, F, H, and J (Bedford limestone) have been plotted as mechanical-value diagrams. Note how the E. U. per foot-pound applied on 100 g. decrease consistently from coarse to fine: H, 0.375; J, 0.238; E, 0.051; and F, 0.011. Similarly in Fig. 19, K, L, M, and N are plotted to E. U. diagrams with results decreasing consistently from coarse to fine: K, 0.135; L, 0.071; M, 0.0544; and N, 0.0372 E. U. per foot-pound-applied on 100 g.

27. On another ore, an Arizona porphyry, a series of crushings similar to K, L, M, and N, gave results from coarse to fine as follows: O, 0.055; P, 0.049; Q, 0.0296; R, 0.0149; S, 0.0127; all E. U. per foot-pound-applied on 100 g. The range of sizes was small; the initial feed averaged

TABLE I.—*Summary of Results*

Character of Rock	Mexican Siliceous Ore ¼ in.		Bedford Limestone									Porphyry, Mill Feed				
Operation	A	B	E	F	G	H	J	K	L	M	N	O	P	Q	R	S
Mesh-Grams of Surface Produced (Plus 350 Reciprocal)	1,400	6,900	8,800	8,450	19,060	665	3,036	3,110	2,400	1,475	2,050	1,250	1,225	1,470	1,254	1,220
Foot-Pounds Energy Applied	80.5	401	166.3	295.6	461	12.7	93.3	35.2	33.5	20.8	32.1	50	48	56¼	103.3	75
Mesh-Grams per Foot-Pound (Plus 350 Reciprocal)	**17.3**	**17.21**	**53**	**28.7**	**41.3**	**52.5**	**32.6**	**88.4**	**71.9**	**70.8**	**63.8**	**25**	**25.5**	**26.1**	**12.15**	**16.3**
Stadler's E. U.'s per Foot-Pound Applied on 100 g	0.041	0.018	0.051	0.011	0.213	0.375	0.238	0.135	0.071	0.0544	0.0372	0.055	0.049	0.0296	0.0149	0.0127
Remarks	Series Coarse	Fine Graduated Crushing	Medium	Fine Choke Crushing	Large Block	Coarse Pieces	Choke Crushing	Series Coarse	Medium		Fine	Series Coarse		Medium	Choke Crushing	Fine

about 4 mesh, while all five crushings resulted in the production of not more than 8 per cent. of −200-mesh sieve product. This work was done in December, 1913.

28. Fig. 20 shows the results of plotting an approximate average of feed and discharge for all tests against both the E. U. per foot-pound-applied on 100 g. and the mesh-grams per foot-pound (+350 reciprocal).

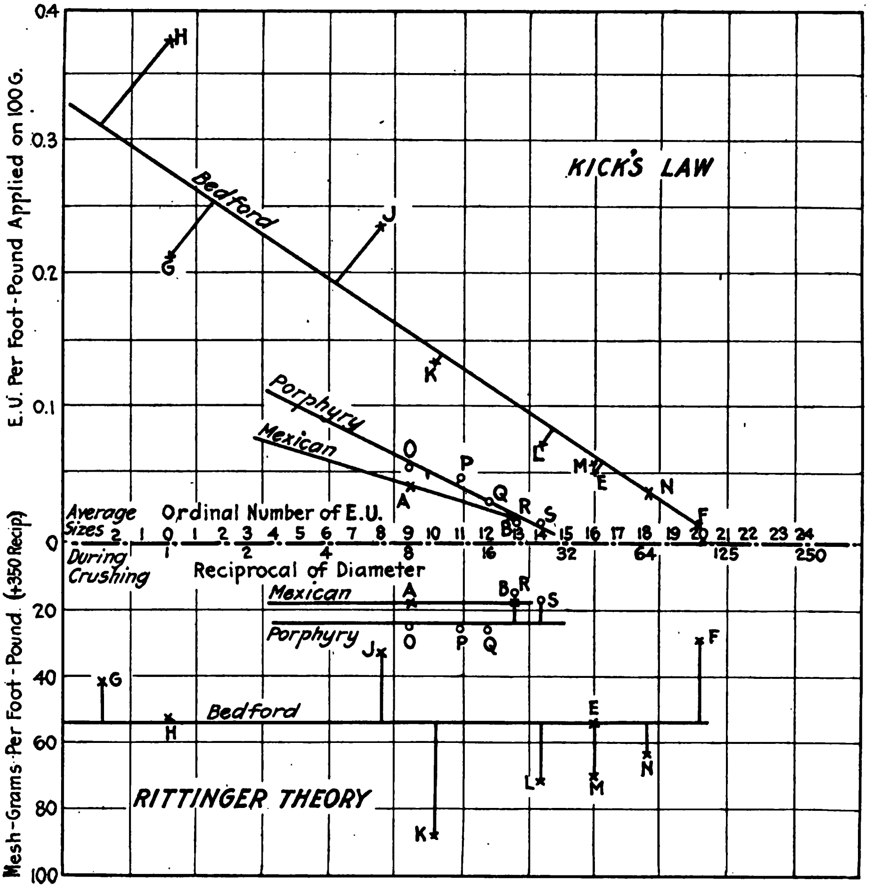


FIG. 20.—VARIATION OF TEST RESULTS BY BOTH METHODS.

The estimated average, taken as near to the center of gravity of the mechanical-value diagrams as possible by ocular observation, gave the horizontal position of each point, while the calculated E. U. and mesh-grams gave the two vertical positions.

In the upper diagram the inclined line (Bedford limestone) connecting the various points shows that for different average sizes of material undergoing crushing, the production of E. U. per foot-pound-applied on 100 g.

decreases from coarse to fine; according to Stadler's interpretation of Kick's law there should be no decrease, they should be the same at all sizes. From the values tabulated in Table I, which is a summary of all these tests and calculated results, it will be found that for the Bedford limestone, the *coarsest* crushing gave 34 *times* the number of E. U. per foot-pound-applied on 100 g. that the *finest* crushing gave. For the other materials, the graphs incline similarly and in sufficient degree to show the fallacy of Stadler's application of Kick's law.

29. In the lower diagram, the results of this work on a Rittinger basis have been plotted, adhering, however, to the average sizes, taken as described above, which will not affect the results, except to lengthen the curve in the coarse sizes and to shorten it in the fine sizes. Horizontal straight lines averaging the results for each material have been drawn in; if the same number of foot-pounds-applied produce the same number of mesh-grams at all sizes, then the Rittinger theory is correct, and a horizontal line represents it in this diagram. If some of the discrepancies be explained as due to "choke crushing," producing an excess of unmeasurable −200 mesh (−350 reciprocal), the majority of points will come nearer to a horizontal line, and better confirm our belief in the Rittinger theory. The results clearly show that the Rittinger theory is so much more nearly correct for crushing operations, that I believe we can accept it and use it for the determination of crushing constants, for machine comparisons, for efficiency tests, etc.

30. Since it is obvious that the sizes finer than our ordinary screens will separate are important if the Rittinger theory be correct, finer screens or classification followed by microscopic measurements will probably be necessary for accurate determinations. To some extent at least, we can approximate the quantity of the finer sizes; there seems to be a law connecting the quantities of different sizes, that the discharge line of a crushing-surface diagram from a uniform crushing operation will be in the form of a hyperbola, and, when plotted on logarithmic paper, as in Fig. 21, will appear as a straight line. In Fig. 21 the product line of N indicates that it would have some 4 per cent. of 1,000-reciprocal size and some 0.6 per cent. of 10,000-reciprocal size; they have surfaces respectively equivalent to 40 per cent. of 100 reciprocal, and 60 per cent. of 100 reciprocal, taken separately; and if these values are correct, these sizes are fully as important in the consideration of energy disposal as all of the +100-reciprocal material. The law of the finer sizes might be something of this form:

$$RW^x = C$$

R being the reciprocal of the diameter (theoretical mesh) under consideration; W, the weight of the material passing through a screen of this diameter opening; and x, a constant depending upon the character of the

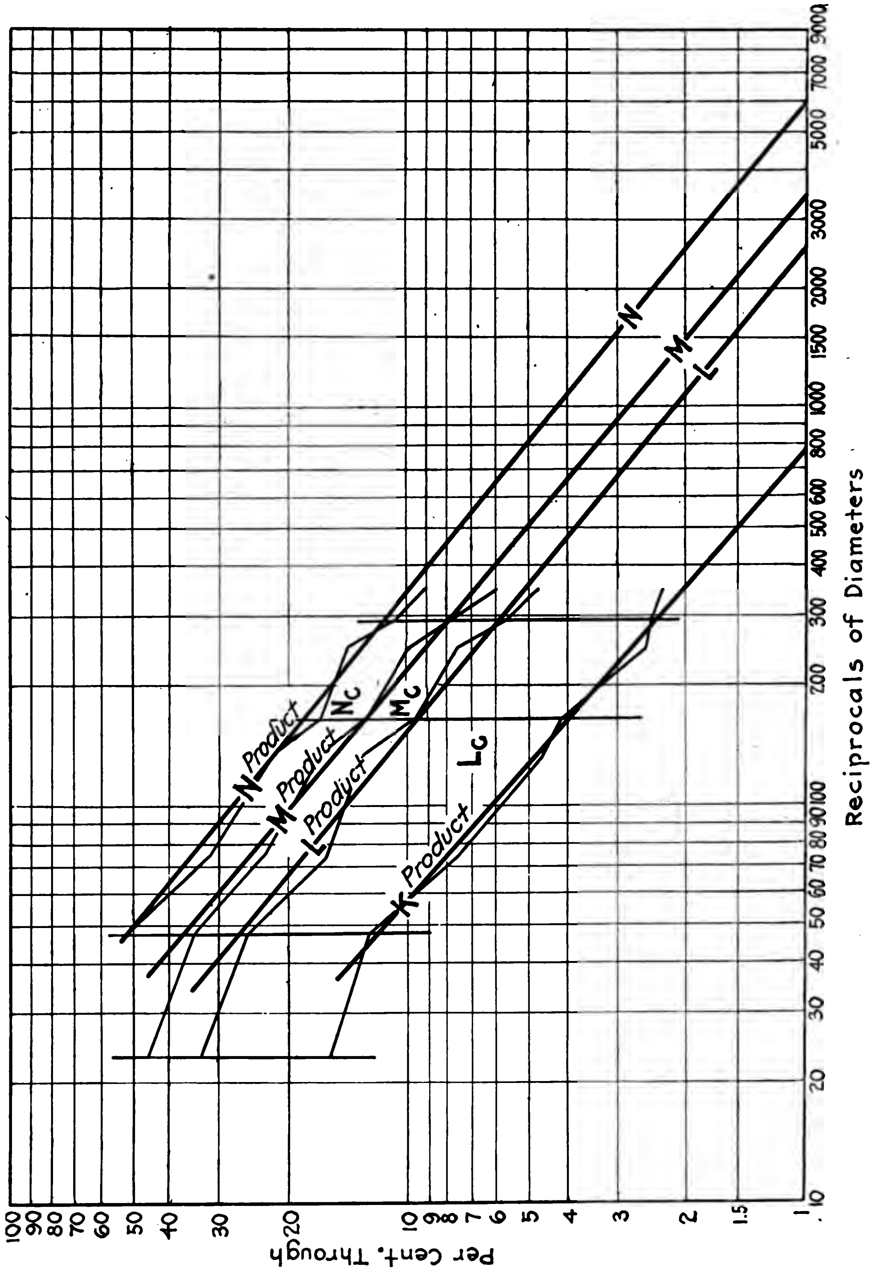


Fig. 21.—Screen Analyses Plotted to Logarithms.

ore, the type of machine in which crushing is done, and possibly upon the screen analysis of the feed; and C, the number of crushing units (mesh-grams, mesh-tons, etc.) in the undersize, considered as all of that one size, and proportional to the area between the co-ordinates of the point on the curve (intersection of C and W) and the zero lines.

This, however, is largely speculative, and is only suggested because it may be of some value to some future investigator.

SUMMARY OF RESULTS

31. Table I gives a summary of results. It will be noted that in most cases the high values for mesh-grams per foot-pound were obtained with low values in foot-pounds, and *vice versa.* This seems to indicate that the highest values and therefore the most efficient crushing would be obtained when each particle was just cracked; a thing that is generally understood to be true in connection with rolls, for example, free crushing being, within limitations, more efficient than choke crushing. Choke crushing here gave uniformly low results, as might be expected. In the case of B, the several crushing operations involved small amounts of power, but they were lumped together for final screen analysis. If all the power had been applied at one time, without intermediate screening, there would not have been such a nice agreement in the mesh-grams per foot-pound, between A and B.

32. As the writer sees the situation, after the acceptance of the substantially correct theory of crushing, there must be recognition of the fact that rocks of varying characteristics are met in crushing operations, that these rocks offer different resistances to crushing, and that it is unfair to compare the operations of two crushing machines upon dissimilar ores (even if they look alike) until some values are given to the resistance or energy absorption of these rocks under crushing conditions. Probably these units will be determined practically by applying a definite amount of energy to a definite weight of the rock, all of a predetermined standard size. The work of determining these constants should be done thoroughly by competent hands. The matter of checking my conclusions can be accomplished perhaps in several of the school testing laboratories of the country. No elaborate apparatus, and apparently no very high degree of accuracy, will be required to prove that Kick's law is of no use to mill operators and engineers.

33. This paper has been prepared under some disadvantages which may account for the possible lack of logical arrangement. The matter was originally submitted to the committee of the Institute in May, 1914, and upon their suggestion the paper was enlarged, corrected, and rearranged. The writer acknowledges his indebtedness to their suggestions. Acknowledgment should also be made of the courtesies extended by

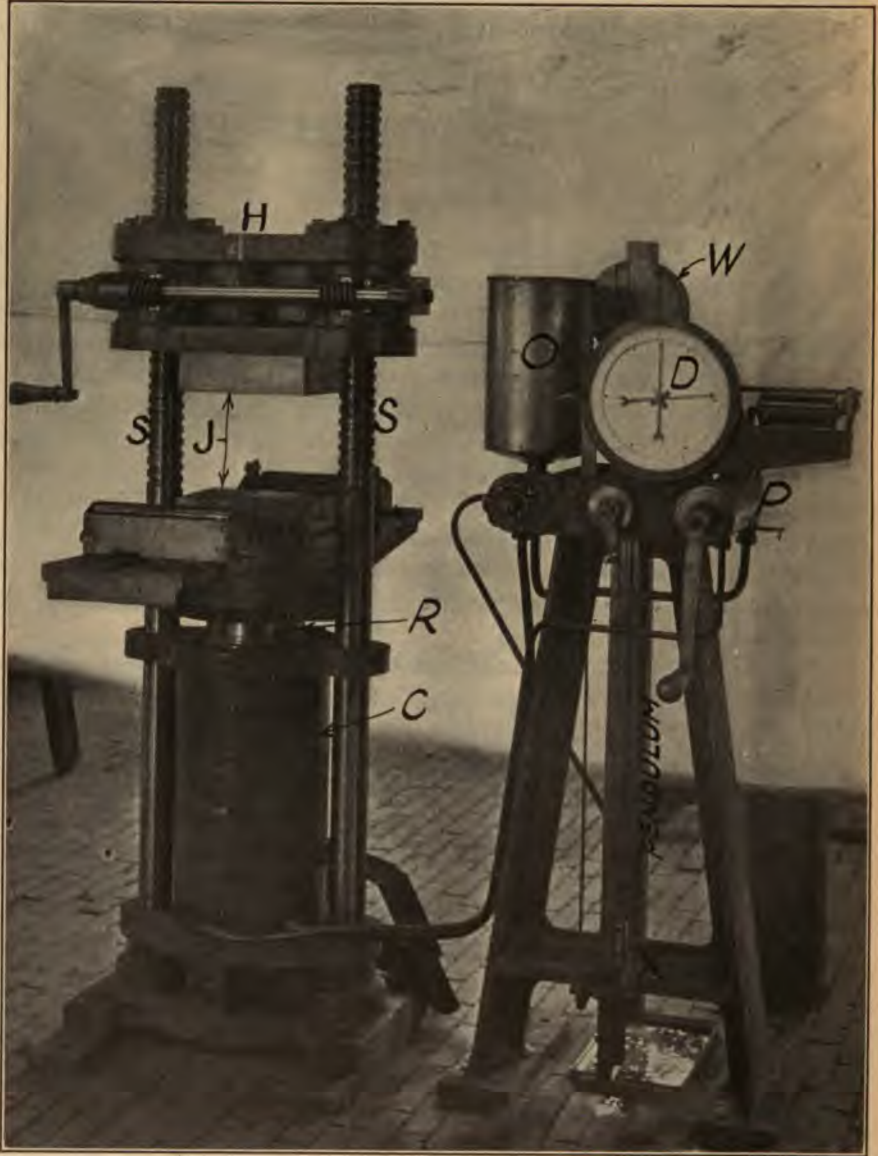


FIG. 22.—AMSLER-LAFFON TESTING MACHINE.

Dr. W. K. Hatt, head of the department of Civil Engineering, and Professor Schofield, at Purdue University, who allowed the writer free run of the laboratories for Testing Materials.

APPENDIX

The Amsler-Laffon hydraulic testing machine, with which these experiments were made, is shown in Fig. 22.

The material is crushed between the faces of the jaws *J* by pressure applied against the ram *R*, contained in the cylinder *C*. The head *H* carrying the upper jaw can be moved up and down the fixed screws *S* by nuts having worm-wheel threads cut on their outer surface into which mesh two worms driven by the hand crank shown to the left.

The pump *P* for supplying the pressure to the ram cylinder *C* is operated by a hand screw on the side of the frame carrying the registering and auxiliary mechanism. Suitable piping connects the pump with the cylinder, the supply tank *O* and the registering cylinder *X*. From *X* suitable links and levers connect to the pendulum and adjustable weight *W*, the swing of the pendulum, which is proportional to pressure exerted, being recorded on the drum or indicated upon a dial *D*. In all of this work the readings of the dial were used. With different settings of the weight *W*, four different scales of pressures having maximum values of 15,000, 50,000, 100,000 and 150,000 lb. may be used, suitable dials being provided.

The deflectometer, which is a part of the regular laboratory equipment, consists of a lever carried at its fulcrum in a suitable base. At the short end of the lever a fine-thread vertical adjusting screw can be brought into contact with the upper jaw *J* of the machine, while the long end of the lever fits against a vertical scale, and moves a vernier with it as it moves up and down, thus permitting the measurement to 0.001 in. of the movement between jaws.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

Standardizing Rock Crushing Tests

BY MYRON K. RODGERS,* B. S., LOS ANGELES, CAL.

(San Francisco Meeting, September, 1915)

In rock- or ore-crushing tests all data, in order to be valuable for study and comparison, should be obtained and tabulated under conditions as uniform as possible. The results of many such tests have no value for comparison, because of incomplete data and the lack of uniform conditions, screens and scale of sizing, etc.

The purpose of this paper is to suggest a form of standard specifications which may be developed by the Milling Committee and approved by the A. I. M. E. for rock-crushing tests. It is not intended to discuss any theories of rock crushing.

Reports of rock-crushing tests should include the following details:

1. Description of the machine employed (jaw or gyratory crushers, rolls, stamps, tube mills, Chilean mills, etc.).
2. Method and material (timber, concrete, etc.) of foundation. Much power is dissipated in the vibration of poor foundations.
3. Locality from which the rock or ore was obtained, and geological, mineralogical, and physical characters of the material.
4. The power consumption of the machine running with no load and with full load, the unit of power being 1 hp. per 24 hr.
5. The capacity of the machine in tons (of 2000 lb.) per 24 hr.
6. The duty in tons per horsepower-day.
7. The screen analysis of feed and product by the proposed A. I. M. E. standard screen scale.

On the foregoing statement, the following remarks are offered.

Toughness of Rock

Table I shows the relative crushing duty of the same machine on three different characters of rock, all other crushing conditions being the same.

This table indicates that the specific resistance of the rock to crushing, may affect the crushing duty of a unit of power as much as 300 per cent. Perhaps this quality may be called toughness. It is not a function of hardness only. For comparison of tests, rocks should be divided into

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*Mining Engineer.

TABLE I.—*Relative Crushing Duty of the same Rock-crushing Machine on Three Different Characters of Rock, all Other Crushing Conditions Being the Same*

Rock and Feed	Mill	Per Cent. Solids	H.p. of Machine	Tons per Machine, 24 hr.	Tons per Hp.-Day	Tons 200 Mesh per Hp.-Day	Screen Scale +4+10+20+40+60+100+200−200								
Calumet & Hecla conglomerate, ¼ mesh.	Conical tube 8 ft. by 30 in.	40	50.0	65	1.30	0.44	Feed	0	65	29	5	1	..	..	..
							Product	..	..	..	3	9	13	41	34
Cœurd'Alêne quartzite, F. M. & S. Co., 5 mm.	Conical tube 8 ft by 22 in. 28 r.p.m.	43	47.5	112	2.50	0.67	Feed	0	43	43	11	1	1	1	1
							Product	..	..	2	22	18	19	12	27
Miami, ¼ mesh.	Conical tube 8 ft. by 22 in.	27	47.5	229	4.85	1.80	Feed	4	45	33	10	2	2	1	7
							Product	..	..	2	14	12	17	14	38

Relative crushing duty of 1 hp. due to character of ore only:

Calumet & Hecla Conglomerate	1.00
Cœur d'Alêne Quartzite	1.92
Miami	3.75

at least three classes in this respect; and the class to which a particular rock belongs might be determined by its crushing strength per square inch.

Screens

For screen analysis, a standard screen scale should be used, having a uniform ratio between the different sizes.

The results of an investigation made by the U. S. Bureau of Standards, with several of the large mining companies, where 240- to 260-mesh screens are used in connection with flotation tests, and including several cement manufacturers and screen manufacturers, are that a screen analysis for all rock-crushing tests, and practically all testing of materials, will be covered by a screen scale based on the U. S. Bureau of Standards, 200-mesh screen with a square opening of 0.0029 in. and a wire of 0.0021 in. diameter, advancing with a ratio of 1.414 (*i.e.*, the square root of 2, suggested by Rittinger) between the successive sizes of screens, to a square opening of 4.197 in. area corresponding to a 4-mesh screen; and where finer than 200 mesh is required, a 280 mesh or 0.0020 opening could be used, retaining the above ratio.

TABLE II.—*Standard Screen Scale*

Mesh per Linear Inch	Opening, Inches	Opening, Millimeters	Diameter of Wire, Inches	Diameter of Wire, Millimeters	Area of Openings, Square Inches
	4.20	106.60	0.375	9.52	17.64
	2.97	75.39	0.207	5.26	8.82
	2.10	53.33	0.192	4.88	4.41
	1.49	37.73	0.149	3.78	2.20
	1.05	26.67	0.149	3.78	1.10
	0.742	18.85	0.135	3.43	0.551
	0.525	13.33	0.105	2.67	0.276
	0.371	9.423	0.092	2.34	0.138
3	0.263	6.680	0.070	1.78	0.069
4	0.185	4.699	0.065	1.65	0.034
6	0.131	3.327	0.036	0.91	0.017
8	0.093	2.362	0.032	0.81	0.0086
10	0.065	1.651	0.035	0.89	0.0042
14	0.046	1.168	0.025	0.64	0.0021
20	0.0328	0.833	0.0172	0.44	0.00108
28	0.0232	0.589	0.0125	0.32	0.00054
35	0.0164	0.417	0.0122	0.31	0.00027
48	0.0116	0.295	0.0092	0.23	0.000135
65	0.0082	0.208	0.0072	0.18	0.0000672
100	0.0058	0.147	0.0042	0.11	0.0000336
150	0.0041	0.104	0.0026	0.07	0.0000168
200	0.0029	0.074	0.0021	0.05	0.0000084

If the analysis is to be carried finer than 0.0029 in. (200 mesh), the next finer sieve-opening in the Screen Scale Series, is 0.002 in. (280 mesh).

The writer is under obligations for practical assistance in the preparation of this paper to W. S. Stratton, Director, Rudolph J. Wig, and R. Y. Turner, of the U. S. Bureau of Standards; Frederick Laist, Metallurgical Engineer, and Albert E. Wiggin, Superintendent of Concentration, of the Anaconda Copper Mining Co.,; and A. S. Disbro of the W. S. Tyler Co., Cleveland, Ohio. Mr. Disbro has practically developed the suggested screen scale.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
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The Rôle and Fate of the Connate Water in Oil and Gas Sands

Continued discussion of the paper of Roswell H. Johnson, presented at the New York meeting, February, 1915, and printed in *Bulletin* No. 98, February, 1915, pp. 221 to 226. See also *Bulletin* No. 101, May, 1915, pp. 1157 to 1162, and No. 103, July, 1915, pp. 1449 to 1459.

C. W. Washburne, New York, N. Y. (communication to the Secretary*).—Mr. Johnson recognizes that connate waters must be driven outward toward the earth's surface. He considers the ascent due to the settling and packing of the underlying sediments and to the formation of gas within the rocks, supplemented by abyssal pressures.

The origin of the methane in the sedimentary strata probably is not simple. There are three possible modes of origin: (1) as marsh gas formed near the surface and entrapped in the sediments; (2) from abyssal sources; (3) by the decomposition of organic matter buried in the sediments. The abyssal source is generally negligible, and in all cases doubtful. The third origin is the most important in the gas fields. A fourth mode of origin, by the dissociation of liquid hydrocarbons, frequently has been suggested, but it is improbable, because crude oils are exothermic. Therefore they have no tendency to split or decompose at low temperatures, as do endothermic substances, such as coal, which is altered geologcally by the loss of methane, water, and carbon dioxide. This cannot happen to a hydrocarbon until its temperature of dissociation ("cracking point") is reached, which is over 250° C. for the most easily cracked hydrocarbons and over 400° C. for others.

Engler says that the dissociation of fats and waxes into oils is an endothermic process, thereby unconsciously nullifying his own theory of the dissociation of these substances at low temperatures. However, recently I have found reason to think that heat is liberated by the disruption of the molecules of fats and waxes, as it is in the molecular decomposition of the substances of wood, coal, etc. If this tentative opinion is correct, and Engler wrong, then it seems that the Engler-Höfer hypothesis of the origin of oil becomes not only possible, but even probable.

A second conclusion is that all complex organic molecules which remain in their original state to the extent of retaining combined oxygen must have a tendency to decompose, with the production of exothermic substances, including methane and other hydrocarbons. This would result in pressure, since the products occupy greater volume than the parent substances. To reverse the problem, as in the paper, and to say that pressure itself would cause the production of methane from organic matter, is a confusion of cause and effect, and seems to be contrary to physical principles. Those who appeal to pressure in this way probably are overlooking the essential factors. Since the investigations of Brame and Cowan[1] and of others have shown that coal is endothermic, its natural underground alteration, even at low temperatures, is explained readily by the common principles of thermo-chemistry. The expression "dynamo-chemical stage" of alteration could be replaced advantageously by the better established word, "thermo-chemical."

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* Received July 6, 1915.

Abandoning pressure as a cause of the liberation of methane from organic matter does not invalidate Johnson's main conclusion that the formation of methane from entombed organic matter probably increases with depth, because a downward increase would result from the increase of temperature with depth, and also from the greater age of the lower strata, in which the process has been longer in operation.

Naturally I agree with Johnson that the formation of methane crowds away an equivalent volume of water, which must move upward toward the surface because there is no other direction in which escape is possible. The water squeezed out by the settling, consolidation, and cementation of the sediments also must escape upward. Locally also there is reason to believe in the ascent of abyssal fluids, and the pressure of the latter probably supplements that arising from the formation of methane and other gases. I had independently reached the conclusion that there is a general ascent of rock fluids across the strata, from a study of the general distribution of gas pressures, and from the abnormally high chlorine content of oil-field waters.

The evidence of this ascent and the reasons for believing that it is a process in general operation cast doubt on the existence of true connate water in any stratum. The connate water would be shoved out or modified by the entry of other water from below, and rarely could any water strictly be connate to the strata in which it is found. Originally it may have been connate with some lower strata, which in the Appalachian fields would be in early or pre-Paleozoic formations, which were laid down at a time when the sea was comparatively young, and when theoretically the sea-water ratio of Cl:Na may have been as high as that in the waters of deep wells. To some geologists this explanation will be more acceptable than the alternative, that juvenile water rich in calcium and magnesium chloride has entered generally into the strata of oil fields and in many other places; but whatever explanation of this is preferred, the conclusion remains inevitable from any point of view, that water and probably other rock fluids have risen far across the strata. Probably there is no free water left in the older strata that strictly is connate with these formations.

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[1] *Journal of the Society of Chemical Industry*, vol. xxii, No. 22, p. 1230 (Nov. 30, 1903).

In the paper on the capillary concentration of gas and oil, I avoided reference to the possible influence of viscosity in promoting the concentration of oil in the larger pores, because it would operate against the apparently identical segregation of gas. Moreover, I have come to think that most of the migration of oil in fine rock pores takes place when the oil is still comparatively light. (I hope soon to present the geological and chemical reasons for believing that all crude oils are descended from lighter parent oils.) At the temperatures prevailing at depths of 2,000 m. light oils have approximately the same viscosity as water, some even less, so that viscosity cannot be an important factor in the concentration or segregation of such oil in its original migration. In the later and comparatively shallow migrations of the condensed heavier oils, such as a typical fuel oil of California, viscosity would be a determining factor in causing the oil from fissures to enter sandstone rather than shale. In fact, such oil could not enter even dry shale, and no oil could enter wet shale from fissures if the conclusions from capillarity are correct. However, fissure migration is a secondary feature, although of great local importance in the formation of some "pools." The original concentration of the oil in sands and fissures must have been due to capillary action operating on comparatively light oils. There is no other way by which the oil of the California fields could have been removed so nearly completely from the diatomaceous and foraminiferal shales in which it is thought to have originated. If the oil were then in its present viscous, highly asphaltic condition, only a very small part of the oil could have been removed and the shales would be generally petroliferous and dark colored, instead of being very locally saturated and generally light colored, as they actually are. In other words, I accept Arnold's conclusion that the oil of California gets heavier as it migrates, and I believe also during its storage in the sands, which is in accord with all sound observation and theory, and I oppose the view of Höfer that crude oil grows lighter with time.

Johnson's three final conclusions seem more than the evidence justifies. We know only that the deep sands of the Appalachian and a few other fields generally are dry. We do not know that the shales are dry. In fact, I suspect that the shales are moist, since the sands still contain water in places. Hence it does not follow that the very deep oil of such fields is scattered in the shale, largely beyond recovery. The value of the very deep reserves is diminished, *first,* by the expense of deep drilling, and *secondly,* by the lack of sufficient water in the sands to control the accumulation of the oil in definite structures. The few definitely synclinal fields, all in dry sands, as near Bluff, Utah, also in southwestern Wyoming and southern Kentucky, have not been very successful. To produce a profitable accumulation of oil *in a continuous sand,* the sand must contain enough water to greatly restrict the oil-bearing area.

Deep drilling for oil admittedly is a failure in the Appalachian (or

Eastern) fields of the United States; but in California, where the deep sands contain much more water, there is good hope of developing very deep fields. Even now there are a few California wells producing oil from below 4,000 ft.

The two remaining conclusions of the author rest on similar evidence and are subject to the same criticism. The general existence of large deep reserves of natural gas cannot be predicted from the available evidence. Possibly the deep gas may be badly contaminated with abyssal nitrogen, as appears to be the case in the western fields of Kansas. The peridotite dikes of western and of central Kentucky, and the more widely distributed barite-fluorite deposits of the same regions, indicate that abyssal emanations may be important in the deep strata of the sub-Appalachian oil and gas belts. These are interesting possibilities. The evidence to be obtained in deep wells offers great promise of contributions to geological knowledge, in fundamental problems that have received little attention. However, there is little justification for general economic predictions concerning the zone immediately beneath that now explored by the drill.

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Rotary Kilns for Desulphurization and Agglomeration

BY SAMUEL E. DOAK, E. M., PHILADELPHIA, PA.

(San Francisco Meeting, September, 1915)

THE utilization of rotary kilns, of the well-known cement type, for the preparation of iron ores for the blast furnace, has become of considerable economic importance within the past 10 years in certain localities, notably those where the native ores are lean. Such kilns may be used simply for the agglomeration of ores too fine to be charged into the blast furnace (*i.e.*, ores whose loss in the dust catchers is excessive, and whose fineness is detrimental to the working of the furnace), or for the desulphurization of lump and fine ores the sulphur content of which does not exceed 6 or 7 per cent. The desulphurizing action can be combined with that of agglomeration provided the kiln is sufficiently large. Other processes have been developed for the same purpose, but while I am not able to draw a comparison between them by reason of lack of sufficient data, it is safe to say that the kiln, if properly dimensioned for the character of ore to be treated, will compare favorably in its results, both as to economy and quality of product, with any other process. It is not, however, as well adapted for the treatment of flue dust as is a sintering operation, on account of the large percentage of fine coke contained therein. Any combustible entering the kiln at its feed end renders the process very much more expensive and unsatisfactory on account of the difficulty of controlling the temperature, also because of the rapidity with which long "rings" form, due to the increased heat at the cooler end of the kiln.

The greatest value of the kiln at present is in the treatment of pyrites cinder; that is, the residue from pyrites ore after the extraction of the greater part of the sulphur. This cinder, while high in iron content, has too much sulphur remaining in it to be allowed to form a large part of the blast-furnace burden, without further treatment. It will range in metallic iron from 42 per cent., representing a very low-grade cinder from a domestic pyrites, to 64 per cent. of the residue from the best Spanish ore, and the corresponding silica content will vary from the maximum of about 25 per cent. to 1½ per cent. The residual sulphur depends largely on the type of burner used in the original desulphurization. In general, the leanest original ores retain the greatest amounts of sulphur. In modern practice the percentage of sulphur in the cinder is rarely as great as six, and seldom lower than one and one-half.

Average analyses for iron, silica, and sulphur of the cinders that have come under my personal observation are as follows:

	Iron, Per Cent.	Silica, Per Cent.	Sulphur, Per Cent.
Virginia fines	48	18	4.2
Virginia lump	54	12	5.0
Spanish fines	58	8	2.5
Spanish lump	63	2	2.0

In addition, nearly all cinders contain zinc, copper (frequently in sufficient amounts to be extracted commercially as a by-product), calcium, magnesium, etc., and sometimes lead. The total of these accompanying elements, however, is usually well under 6 per cent.

The process is a simple roasting, with the addition in the case of fine ores of a nodulizing or agglomerating operation, performed near the discharge end of the kiln, at a higher temperature than is necessary for the desulphurization. The sulphur is combined in the ore as sulphide and sulphate of the various bases, and if the kiln is not forced over its capacity there is no difficulty in reducing it to less than 0.05 per cent. of the finished product. As far as the desulphurizing action is concerned, the principal caution to be observed is not to allow the kiln to get too hot at the back end. To get a dead roast the operation should be very gradual, and the heat should range from a barely perceptible dull red at the feed end to a bright yellow at the discharge end. When the ore is too hot in the center of the kiln, there seems to be a tendency to hinder the desulphurization, probably on account of an almost imperceptible glazing of the particles of cinder, that renders them impervious. For this reason it is important that the roasting should be complete before the nodulizing begins, since for the success of the latter operation the ore should be sufficiently pasty to ball up. This pastiness, however, should never be carried to the point where there is any flow of the cinder, and in fact it should never be visible in the kiln. A bright yellow heat is sufficient for the nodulizing, and it is well to keep the zone of agglomeration as near the discharge end of the kiln as possible, 15 ft. being sufficient length for the operation in any kiln not more than 100 ft. in length. For this reason a short flame is very desirable.

Oxygen necessary for the desulphurization is furnished either by stack draft at the feed end, by a blower at the discharge end (in connection with the fuel supply), or by both. The height of the stack is determined more by the necessity of raising the sulphur fumes above the height where they are obnoxious to people, and harmful to vegetation, than by the oxygen requirements of the process, as the slope of the kiln together with the draft induced by the coal blower is sufficient in small kilns to furnish the required amount of air without a stack. A kiln 60 by 5 ft. was run for several months without any stack, with excellent results, the only artificial draft being supplied with the powdered coal by an Aero pulverizer.

This same kiln on an 80 by 4 ft. stack gave incomplete desulphurization and an excessive amount of "ring" trouble, due to too high a temperature at the feed end.

At present the rotary kiln owes what popularity it possesses to its nodulizing feature, in spite of the fact that it is a cheap and efficient roaster, and that it is much more successful, as a roaster, working on coarse than on fine ore. As I remarked above, the sulphur should be practically eliminated from the ore before nodulizing begins. The most satisfactory way of insuring this would be to separate the two processes by using two kilns, but that would be impracticable except in a very large plant, and as a matter of fact the whole operation can be done very well in one kiln provided it is not too small. The greatest advantage of separating the processes would be that "ringing" could be almost entirely eliminated in the desulphurizing kilns, and confined to the nodulizing kilns, which could be made short and of large diameter and great capacity.

It has been the custom arbitrarily to connect a certain diameter with a certain length of kiln. A 60-ft. kiln is usually 5 ft. in diameter; 80- and 85-ft. kilns are 6 or 7 ft. in diameter, and a 100-ft. kiln is ordinarily 8 ft. in diameter, but it would frequently be more economical to proportion the kiln in a less haphazard way than by accepting the builder's standard designs. The two determining factors should be the output desired, and the quality of ore to be roasted. The output is dependent on the dimensions of a cross-section of the bed of ore that can be carried in the kiln, the slope of the kiln, and its speed of revolution. The slope of the kiln, however, can be left out of consideration. Good practice places it at ¾ in. to the foot, and increased or decreased output can be better obtained by changing either of the other variables than by changing this. Speed control, too, is a feature of the design of every kiln now, so that this factor may be adjusted to the most satisfactory point after the kiln is in operation. The question of output, then, so far as the kiln design is concerned, is determined by the width and depth of the ore bed. In general, the latter can be increased with the length of the kiln, since the length of time it takes the ore to pass through the kiln will be greater as the kiln grows longer, all other things being equal, but the increased capacity due to the length will not be very great, since, without changing the diameter, the ore bed cannot be deepened to any great extent without harming the roasting process, for which a continual and complete stirring and overturning of the ore bed is necessary. Too great a depth of bed prevents this.

An increase in the diameter of the kiln, on the other hand, increases the possibilities of the output considerably, by allowing the cross-section to be made much greater with a slight increase in depth, giving the desired effect of a greater ratio of surface of ore exposed to the air in comparison with the area of the cross-section. It would seem reasonable, then, to base the length of the kiln entirely upon the time necessary for desulphurization and agglomeration, and to control the output by the diameter.

The question of the speed of progression of the ore in the kiln, upon which, together with the kiln length, depends the time available for treatment, cannot be accurately determined beforehand, as it is controlled by too many variables. The size of the ore particles, and their shape, the depth of bed, condition of the lining, and the size and shape of the ring all have an effect on the speed of progression of the charge. Fortunately, however, we know from actual practice that any type of cinder can be treated satisfactorily in kilns 80 to 100 ft. in length, turning at a speed of from one revolution in 60 sec. to one in 90 sec. The greater length is safer in treating all fine, high-sulphur cinders, since it reduces greatly the chances of a poor product through careless supervision.

The tonnage output of a kiln varies greatly with the kind of cinder charged. A kiln 60 by 5 ft. with 4½-in. lining will treat 1,500 tons a month of low-sulphur lump ore, while on high-sulphur fines, it will with difficulty take care of 750 tons, and frequently even at this low tonnage a large percentage of the product will not be nodulized. A kiln 85 by 7 ft. will handle 3,000 tons of high-grade cinder per month, and about 1,500 tons of the poorest grades, while a kiln 100 by 8 ft. will vary from 4,500 to 2,500 tons with the quality of cinder treated. A kiln 85 by 8 ft. should handle almost as much as the one 100 ft. long, but at a greater fuel cost. In general, the larger kiln will have the lower fuel cost and the fine domestic ores will need less coal, on account of their higher sulphur content. On a 60 by 5 ft. kiln, the fuel consumption averaged for several months about 12 per cent. by weight of the weight of roasted ore.

The mechanical difficulties that were so common when rotary kilns were first used for the treatment of iron ores have been practically eliminated. The most important things to be observed to insure continuous operation are, to make the kiln sufficiently strong (reinforcing it especially very heavily under the tires), and to keep the operating machinery well protected from the dust. The tires should always be kept tight to the shell, as if any play is allowed the bolt holes will gradually become enlarged, and the kiln will bear down hill so strongly that it will frequently break its thrust bearings. At least an inch of additional metal should be put around the shell under the tire.

The one great obstacle now in the way of continuous operation is the formation of "rings," caused by the cinder becoming hot enough to stick to the lining. As soon as the lining is covered the "ring" increases more rapidly by building up upon itself around an ever-decreasing circumference until it seriously interferes with the operation, and has to be removed. It usually forms in the 15 or 20 ft. nearest the discharge end of the kiln, and puts an excessive amount of weight upon that end. A piece of the "ring" is hard and has the appearance of having been partly fused. It looks very much like mill cinder.

Many devices have been tried to prevent the formation of the "ring," or to remove it without stopping the kiln, but, so far as I know, none of them have been successful. The best plan, in my experience, has been to shut down the kiln, when the ring became too large, cool it off with water, and cut it out with bars and picks. It is impossible, I feel sure, to prevent its formation when running on fine ores. With low-sulphur lump ores that do not need to be nodulized, careful control of the heat will allow a run of indefinite length, but as all the lump cinders (unless screened) carry a considerable percentage of fines (*i.e.*, −20 mesh), sooner or later a shutdown will be necessary. On any kind of cinder, however, the formation of the ring can be postponed by keeping, as nearly as possible, the conditions of temperature, speed, and feed, from fluctuating. The temperature should always be kept as low as is compatible with the successful operation of the kiln.

A 9-in. firebrick is generally used for lining, and is probably the best size on the larger kilns. A 4½-in. brick is satisfactory for kilns of 5 and 6 ft. diameter. The thicker bricks protect the shell better and prevent warping, but in the smaller kilns they reduce the available diameter too much. The lining wears down rapidly, and frequently individual bricks are loosened or broken in cutting out the ring. If the shell becomes warped under the tires, as is frequently the case when the kiln is not reinforced strongly enough at this point, it is very difficult to keep the bricks in. A 9½-in. lining should last over a year in the hot part of the kiln where the ring forms, and very much longer in the cooler end.

Automatic feeders are frequently used to insure a constant rate of charging, but they are really unnecessary, as the slow motion of the kiln will effect this, if only a simple feed pipe is used. All machinery used in handling the ore and the fuel and in rotating the kiln should be as nearly dust proof as possible, and should be driven by fully inclosed motors, as the fine cinder dust is very harmful. The best plan is to keep the feeding apparatus, the machinery for handling the roasted ore, and the fuel pulverizer and blower, all under separate housings. The kiln itself does not need to be housed, except where such housing is necessary to protect its driving machinery. Powdered coal is used as the fuel, so far as I know, exclusively, with the exception of one or two plants where blast-furnace gas has been tried. An oil flame would probably be very satisfactory where cheap oil could be obtained, on account of the ease of controlling the temperature incident to its use.

The cost of production of a ton of roasted ore, aside from the cost of the raw cinder, depends chiefly on the labor cost. Two men on a shift should be able to take care of a well-designed plant producing up to 100 tons in 24 hr., while in a poorly designed plant this number might easily reach four or five. On a small kiln, 60 by 5 ft., the labor cost per ton, on high-sulphur fine cinder, averaged 40c. for several months, and with the same kiln, using high-grade lump cinder, this figure was reduced to 30c., but both these figures were excessive on account of faulty design, that made necessary three and four men on a shift; also on account of a very low output per man, due to the small size of the kiln. On the other hand, this was partly balanced by the fact that wages were low, and only two shifts were necessary. Repairs and supplies should not cost over 8c. per ton of roasted ore on a kiln of any size. The important factor in economic kiln operation is careful designing, and it cannot be said too strongly that the cost of operation of a rotary kiln will be almost entirely determined by the layout of the plant.

The roasted ore, for which the cement term of clinker is used, works very well in the furnace. In Virginia practice, where the blast pressure is comparatively low, it is not essential that the nodules should be large. If 80 per cent. of the clinker stays on a screen of ⅛-in. mesh, there will be no difficulty in the furnace at pressures of under 9 or 10 lb. It is very easy to make larger nodules, but as a higher temperature is required to do so, with the attendant evils of more frequent "ringing," greater fuel expense, and a harder, less porous product, it is better to keep the nodules as small as possible. The clinker frequently forms from one-sixteenth to one-half of the burden without ill effects, working with "brown ore" and specular hematites, and by its use the yield of the Southern furnaces where the native ores are lean can be raised from a scant 42 per cent. to 50 per cent. and over, depending on the quality of the cinder. It is in localities where the native ore is lean that the kiln is valuable. It affords an opportunity of enriching the burden with a high-grade ore at a much lower cost than could be done by direct purchase of lake or other rich ores, and where the silica in the native ore averages well over 20 per cent., it is indispensable that the burden should be sweetened in some way to cut down the excessive amount of flux and consequent fuel necessary to produce good iron.

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The Mexican Oil Fields

BY L. G. HUNTLEY,* PITTSBURGH, PA.

(San Francisco Meeting, September, 1915)

I. HISTORY OF OIL DEVELOPMENT IN MEXICO

THE occurrence of oil or "tar" in Mexico was mentioned as early as the seventeenth century by Friar Sagahun, who gives the Indian name "chapopote," by which these asphalt seepages are still called. This asphalt was apparently used, as it has been used by primitive people in many parts of the world, in religious ceremonies and for medicinal purposes. Travellers also report that the ruins in Yucatan and the pyramids in southern Mexico show traces of the use of "chapopote" as a building cement.[1]

DeGolyer says that the first attempt to exploit oil or gas in Mexico in a commercial way is shown by the records of the Memoria de Fomento of 1865, when permission was granted to a Senor Ildefonso Lopez to exploit the deposits of petroliferous substances in the San Jose de las Rusias area in the State of Tamaulipas. Other concessions follow, and probably refer to surface seepages which occur in that district. This was several years after the discovery of oil in Pennsylvania. In 1868, a well 125 ft. deep was drilled by a company organized in Mexico City, in what is now known as the Furbero district, and a little oil was refined there.

In 1873, residents of Tampico denounced seepages along the Tamesi River, and asphalt was mined near Tempoal in the Canton of Tantoyuca. No drilling was attempted. But between 1880 and 1883 several shallow wells were drilled for oil in Mexico, two of them being near the present Potrero de Llano field west of Tuxpam. The wells were drilled by a Boston company with Canadian rigs, and are said to have reached a depth of about 400 ft., while one is reported to have flowed. Several other abortive attempts were made to drill for oil between 1885 and the beginning of the present century, about which time the Mexican Petroleum & Liquid Fuel Co., Ltd., in which Cecil Rhodes was interested drilled 24 wells unsuccessfully, several of them as deep as 1,500 ft., in the State of Vera Cruz. Also Messrs. Doheny and Canfield of California purchased land in the State of San Luis Potosi, and in May, 1901, the first successful well was struck in the present field of Ebano, 50 miles west of Tampico. Various shallow wells were drilled about this time in the States of Vera Cruz, Tamaulipas, near Guadalupe, Chiapas, and Tabasco on the Isthmus of Tehauntepec; and several asphalt companies attempted to mine the surface deposits in the Cantons of Tuxpam and Ozuluama.

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* Johnson and Huntley, Consulting Oil Geologists.

[1] De Golyer. *Oil and Gas Journal*, April 16, 1914.

In 1904, drilling was commenced by the Oil Fields of Mexico Co. (Pearson interest) at Furbero, and its first well was brought in successfully. Pez No. 1, the first large well on the Ebano property, was also brought in during the same year, and has flowed continuously since. About 1907, drilling became more active, especially in the Cantons of Tuxpam and Ozuluama. It was started at San Diego by the Pennsylvania Oil Co., at Tumbadero by the Pearson interests, and at Juan Casiano and Tres Hermanos (La Pithaya) by the Huasteca Petroleum Co. A Mexican bank drilled four shallow wells near the Tampalache seepage, just north of the Panuco field. Operations were also commenced by Pittsburgh interests at Los Esteros, in the State of Tamaulipas.

In 1908, the Pearson refinery at Minatitlan on the Isthmus of Tehauntepec.commenced operations. In May of the same year San Diego No. 2, the discovery well of the Dos Bocas field, came in with a production of 2,500 bbl. daily. On July 4, the famous San Diego No. 3 was brought in, caught fire immediately, and burned until extinguished by encroaching salt water. The Chijol field in the Ebano district and the Topila field were brought in during this year, which inaugurated important operations in the Mexican oil fields.

The year 1910 saw the discovery well of the Panuco field (East Coast Oil Co. No. 401) drilled, and the first well in the Tanguijo and San Pedro districts were put down in that year by the Pearson interest, the Mexican Eagle Petroleum Co. The Huasteca Petroleum Co. also brought in its No. 6 and No. 7 Juan Casiano, one with 14,000 bbl. and the other with 28,000 bbl. daily production. About the time these came in, this company completed its 8-in. pipe line from that field to Tampico. The Mexican Eagle Co., having completed its pipe line from Furbero to Tuxpam, brought in, in December, 1910, the famous Potrero del Llano well, yielding about 100,000 bbl. daily. Drilling was by this time very active; and in 1911 the first exportations of crude oil were made to the United States from Tampico.

In July, 1912, the Government put a tax of 20 centavos per metric ton (1.1 c. U. S. gold per barrel) upon all oil produced in the Republic. The construction of a large fleet of tank steamers had already been commenced by the Huasteca Petroleum Co. In 1913, the Penn-Mex Fuel Co., now a subsidiary of the Standard Oil Company of New Jersey, through the

South Penn Oil Co., brought in two good wells in the Alamo field near Potrero del Llano. The Chila-Salinas and the Topila field began to suffer severely from the invasion of salt water. The Mexican Eagle Co. brought in the discovery well in the Los Naranjos field in Amatlan, which has since broken its valve and proved to be a much larger producer than was estimated by flowing at the rate of 50,000 bbl. per day for a week. On December 1, the Government raised the production-tax to 75 centavos per ton (approximately 4c. U. S. gold per barrel) besides increasing numerous duties affecting the petroleum industry. A bill was proposed in the Chamber of Deputies for the nationalization of the petroleum industry, but no decisive action was taken.

During January, 1914, the Dutch Shell interest (La Corona Petroleum Company) brought in its big well, which is estimated to have a capacity of 100,000 bbl. per day, in the Panuco field. During the spring of this year the activity of both the Federals and the Constitutionalists became very great in the vicinity of Tampico, culminating in a general exodus of the foreign employees of the oil companies in April, at the time of the Vera Cruz affair. Operations came to a standstill for a time, but the foreign employees gradually returned during the next two months, and restricted drilling was carried on by some of the larger companies, especially in the Panuco field. But continued guerrilla warfare, and the generally hostile attitude of all Mexican factions, led to heavy demands upon all companies, so that affairs in the oil fields were by no means normal when the European war broke out in August. In addition to this, lightning set fire to the big Potrero del Llano No. 4 well, which was not entirely extinguished until after the end of the year. During the latter part of the year hardly a string of tools was running. Gen. Carranza meanwhile appointed a commission to examine into the oil industry in Mexico and to make recommendations as to future methods of regulation. The conditions under which new work might be carried on under this decree were prohibitory, and resulted in a complete shutting down of all work during the early part of 1915. However, the Carranza government later modified its attitude, and agreed to issue permits to companies making application, under which they could operate under certain restrictions, pending the result of the work of the commission. This remains the situation in June, 1915, and development work is being carried on under difficulties, with only about six rigs running. Oil shipments have not been interrupted, however.

II. Production

The following table shows the annual production of petroleum in the Mexican fields since 1904. Within the last three years, this has not represented the productive capacity, which was much greater. The completion of additional transportation lines to seaboard, the building of more tank ships, and the lessening of the over-production of high-grade oil in the United States, together with the broadening market following the end of the European trouble, will see the production of the Mexican fields advance enormously. At present they are yielding only about 70,000 bbl. per day; but the wells already drilled could produce 330,000 bbl. per day. In fact, the latest figures on good authority make this capacity more than 500,000 bbl.

PRODUCTION OF OIL IN MEXICO

Year	Barrels (42 gal.)	Year	Barrels (42 gal.)
1904	200,000	1910	4,099,000
1905	300,000	1911	13,655,488
1906	500,000	1912	16,844,066
1907	1,000,000	1913	24,574,500
1908	3,481,000	1914	25,725,403
1909	2,765,000		

During the latter part of 1914 there were 56 companies actually operating in the Mexican fields. But some 200 companies had been organized and were engaged in the business in one way or another. Most of these were American companies, the exceptions being the Pearson interests (English), the Dutch Shell interest (English and Dutch), and a number of Mexican companies particularly in the Panuco and Topila field. The principal shippers of petroleum were the Mexican Eagle Oil Co., Ltd. (Pearson); the Huasteca Petroleum Co. (Doheny); the Tampico Company; the Mexican Gulf Oil Co. (Mellon); the East Coast Oil Company (Southern Pacific Ry.); the Trans-continental Petroleum Co. (John Hays Hammond); La Corona Petroleum Co. (Dutch-Shell); the Panuco Valley Oilfields Corp., Ltd. (Simms & Bowser); and the Penn-Mex Fuel Co. (Standard Oil Company of New Jersey).

Not more than 50 wells were actually producing, and more than half of the oil produced came from about six of these. The rest were shut in, waiting a better market, or better transportation, or for other reasons more or less connected with business policy, which sometimes included the obtaining of additional territory or the clearing up of titles.

The estimated total daily capacity of all wells in the Mexican fields of 330,000 bbl. may be divided among producing fields as follows, at the beginning of the present year:

Barrels		
150,000	Panuco	Northern district, producing heavy oil, from 10 to 14° Bé.
4,000	Topila	
6,000	Ebano and Chijol	
65,000	Casiano and Los Naranjos	Southern district, producing oil from 18 to 27° Bé.
100,000	Potrero, Alazan, Alamo, Tanguijo, and other points shipping from Tuxpam	

NOTE.—The Huasteca Petroleum Company's production at Cerro Azul and Juan Felipe has not been included through lack of information; but it is claimed that these wells are capable of adding 175,000 bbl. a day to this estimate, making the total 500,000 bbl.

While these figures indicate what might be expected of the Mexican fields if transportation and marketing facilities were to make it possible for all wells to be drawn upon to their full capacity, yet as a matter of fact, the year 1914 saw only an average of 70,000 bbl. per day of actual production distributed as follows:

	Barrels
Topila	398,679
Southern fields	18,830,359
Panuco	5,058,970
Chila-Salinas, Ebano, Chijol, etc	1,108,995
Tehuantepec	328,500
Total Production	25,725,403

This was consumed or stored as follows:

	Barrels
Clearances	20,674,357
Internal consumption	1,172,898
Refinery products	2,000,000
To storage	1,888,148

The addition to stocks brought the total storage in Mexico at the beginning of the year to 15,127,834 bbl. as follows: In steel storage, 9,658,258 bbl.; in earthen storage 5,369,576 bbl. (including 508,690 bbl. unmerchantable).

Clearances from Mexican ports were distributed as follows during 1914:

	Barrels
To the United States	15,476,727
Mexican Coastwise	4,510,061
South America	195,138
To the Continent (Europe)	356,205
To the United Kingdom	69,780
To Panama	46,446
To Cuba	20,000
	20,674,357

It is not to be expected that most of the producing districts in the Mexican fields will decline in a manner similar to the sand pools of the United States. It has been proved that the estimated initial capacity in many wells would be equaled if not surpassed by actual performance if they were allowed to flow freely. In fact, wells such as the Juan Casiano No. 6 and No. 7 and Los Naranjos No. 1 and No. 4, and the Mexican Fuel Oil Company's No. 3 Zurita, all in different districts, increased their production after being allowed to flow, as shown below:

	Initial Production, Barrels	Actual Flow Later, Barrels
Juan Casiano, No. 6	10,000	14,000
Juan Casiano, No. 7	20,000	28,000
Los Naranjos, No. 1	5,000	10,000
Los Naranjos, No. 4	10,000	50,000
Zurita, No. 3 (Panuco)	30,000	80,000

These cannot be considered exceptional cases, since so few of the large wells have ever been allowed to flow freely. Considering geological conditions, which will be discussed later, such behavior is to be expected. The oldest commercial wells in the Mexican fields (Pez No. 1 in the Ebano district, and Juan Casiano No. 7) after producing since 1904 and 1910 respectively, and from widely separated districts have shown practically no decrease in their flow since they were drilled in. The famous Potrero del Llano well, in the southern part of the field, has never produced up to its capacity and is still yielding as much as ever. These last two wells have produced approximately 40,000,000 bbl. of oil each during their history, and show practically no signs of decline, so far as known. The same may be said of the oldest well in the Panuco field (East Coast No. 401), drilled in 1910. This is even more remarkable, since the area surrounding this well has been closely drilled. While a few of the smaller wells at Panuco showed a decline after being shut in for a time, yet this apparently did not indicate any exhaustion, for at the same time No. 3 Zurita increased its production, as indicated above.

It must be said that producing conditions in Mexico differ fundamentally from the fields in the United States, and that, by reason of these conditions there is no vital need on the part of producing companies to bring the oil to the surface as quickly as possible (as at Cushing), and hence relatively little necessity for large tankage—except at terminal stations for operating reasons. The Mexican oil production comes from about 20 different distinct groups of wells. In all except Panuco and Topila, one company usually controls a large acreage surrounding its group of wells. Hence, each of these 20 districts represents a pool, or geological unit, in which only a few initial wells have been drilled. It is thus to the interest of each company to drill efficiently, and to keep wells shut in when there is no market or no transportation. There is usually no waste of oil or gas, except in the case of accidents, or inability to control a well. One of the few cases of waste happened recently, where a company struck a gas well south of the Topila field, which it is allowing to blow wide open, hoping it will show oil. This is almost the ideal condition advocated by some conservationists in this country—one company to each pool.

There has been very little decline of pressure as yet, even in the older pools; and wells such as the first ones at Ebano and Juan Casiano, producing since 1904 and 1910 respectively, are doing so at practically their original rates. In the latter case 40,000,000 bbl. of oil have been taken from the one well, with not more than 10 per cent. decline from its initial daily production. If this locality had been drilled up, as was the case at Cushing, probably it would almost have ceased to produce by this time, and might not have produced as much oil from all wells as will be obtained from the few holes which will be drilled by the present company.

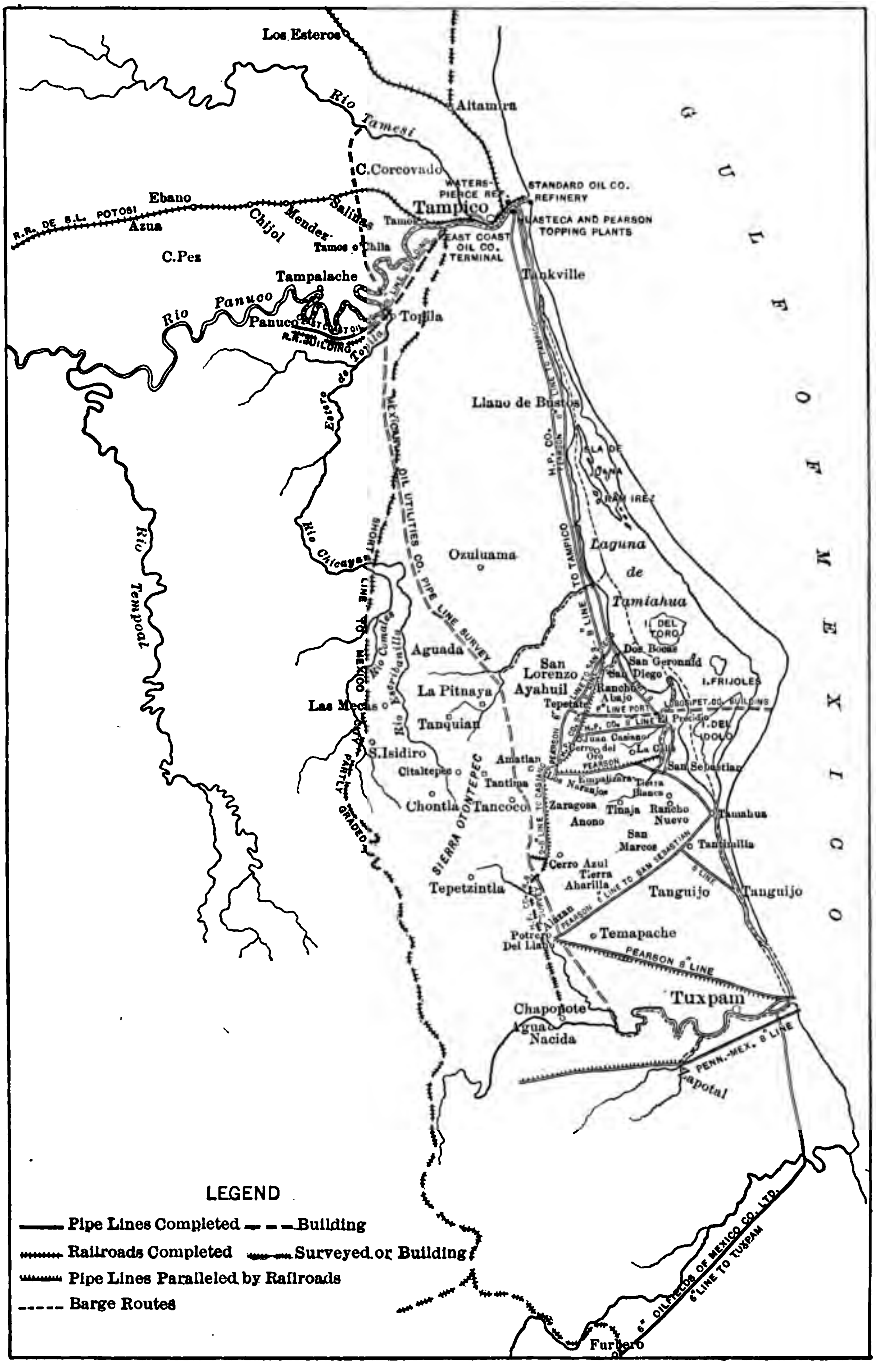


Fig. 1.—Sketch Map of the Mexican Oil Fields, showing Pipe Lines and Railroads.

Transportation.—Oil from the Ebano district is used largely as fuel by the Mexican railways, and considerable asphalt for paving also goes by rail. The Panuco oil all goes in barges to terminals near Tampico, and thence in tank ships to destination. The other fields all pipe their oil to seaboard either at Tuxpam or Tampico, where it is taken by tankers. There are at present 427 miles of pipe line in Mexico, mostly 8 in., some is 6 in. and smaller. About 50 miles additional are building or contemplated. The principal lines are as follows (see Fig. 1):

East Coast Oil Co.; 8-in. line, Panuco-Ponce; about 13.65 miles completed of a 24-mile line to Tampico.

Mexican Eagle Co. (Pearson); 8-in. line, Potrero del Llano to Tuxpam; 8-in. line, Potrero del Llano to Tierra Amarilla, to Tanguijo, to Tamiahua, to San Sebastian, to San Diego, to Bustos, to Tampico; 8-in. Los Naranjos to San Diego. (An allied company has a 6-in. line from Furbero to Tuxpam.)

Huasteca Petroleum Co. (Doheny); two 8-in. lines from Cerro Azul to Juan Casiano; three 8-in. lines from Juan Casiano to San Geronimo, to Esperanza, to Tankville, to Tampico; and a gas line from Juan Casiano to Tampico.

Penn-Mex Fuel Co. (South Penn Oil Co.); 8-in. line from Alamo and Agua Nacida to Tuxpam; 4-in. water line paralleling above.

Transportation is a serious difficulty in getting Mexican oil to the market. Present equipment could not possibly handle the full production of all wells hitherto drilled, which is variously estimated from 330,000 to 500,000 bbl. per day. In 1913, the production of these wells was 24,574,500 bbl. During 1914 none of the old wells declined in production, and 32 new wells came in, representing a possible new production of 196,000 bbl. per day. During the same year a number of pipe lines were enlarged from 6 in. to 8 in., and at least one new field (Los Naranjos) was connected with the seaboard. Shipping facilities from the Panuco field were increased, and some new tankers went into service. In spite of all this new production and new equipment, the field yielded only 25,725,403 bbl. of oil during 1914. At the present time not more than 50 miles of additional pipe line are under construction or contemplated, and very few new tankers, considering this great shut-in production. This indicates that companies are awaiting a better market before making a serious effort to get their product from wells already drilled to points of consumption. A field with a possible daily production of at least 330,000 bbl., which meanwhile only produces 70,000 bbl. per day, and at the same time makes little effort to build new pipe lines and tankers, as indicated by present constructions under way, furnishes strong evidence of the effect of market conditions.

It is to be assumed that development in Mexico will be relatively much slower than would be the case if these fields were located within the United States, for the following reasons:

1. The great expense of operating keeps out all except financially strong companies which will not develop their territory so rapidly as to break the market, and, since they control large areas around their wells can afford to suspend production, awaiting a better market, because they are not losing pressure or oil to their neighbors in the meantime.

2. Political difficulties retard development.

3. Government and State taxes, high-operating and maintenance charges, as well as the cost of transportation by tank ships, add to the cost of the oil delivered to markets such as the United States. These expenses make it impossible to put oil into United States markets for less than 35 c. per barrel at Gulf Coast ports or approximately 50 c. for Atlantic ports. The price for this oil cannot break below these costs for any except small lots, offered at times of a crisis in the field. This is particularly true, since the largest producers also control, either in Mexico or in the United States, their own pipe lines, tankers and refineries.

It may be added that Mexican oil at those prices, plus a profit, must compete with Gulf Coast and California fuel oil, and Oklahoma residues. The fact that it cannot be brought to this country and sold at a price much below these domestic oils prevents its being diverted into as many new channels as usually open when the price breaks to a low point; hence the market will not expand so rapidly. In so far as transportation is a factor of this relatively high price for Mexican oil, it is the reason for this restricted market. And it must be admitted that transportation is thus indirectly the chief factor in keeping down production.

III. OPERATIONS IN DETAIL

The following was the approximate status of all operations in the Mexican fields during the spring of 1915:

MEXICAN EAGLE PETROLEUM COMPANY, LTD. (Pearson interest)

Well No.	Property	Daily Production Barrels	Remarks
1, 2, 3 & 4	Dos Bocas (Lease) Camp abandoned	Small 200,000	Nos. 1, 2, & 3 were small producers. No. 4 came in at 200,000 bbl. a day and burned for 57 days, ruining the field.
1, 2 & 3	San Antonio de Tamihui (Lease)	Dry holes, all more than 3,000 ft. deep	
	Potrero del Llano (Lease)		Wells about 2,000 ft. deep
1		800	Shut in
2		600	Shut in
3		Dry hole	
4		110,000	Gusher
5			Shut down
6			Abandoned
7 & 8			Drilling
	Alazan Ranch (Lease)		Adjoining Potrero del Llano
1 & 5			Abandoned
2 & 3			Salt water
4		11,000	Shut in
6		3,000	Shut in
7		2,000	Shut in
8			Crooked hole
9		3,000	Shut in
10		1,000	Shut in
11, 12 & 13			Drilling
	Tierra Amarilla (Lease)		South Field
1 to 5			Salt water and small amount of oil
6		Can produce 6,000 bbl. oil and 4,000 salt water	Shut in
7, 8, 9, & 10			Producing salt water and small amount of oil.
11 to 15			Drilling or derricks
16			Location

Mexican Eagle Petroleum Company, Ltd. (Pearson interest).—*Continued*

Well No.	Property	Daily Production Barrels	Remarks
...	Tanguijo (Lease)	South field	
1 to 20...	...	Total production about 500 bbl. a day. Nos. 2, 5, 6, 7, 8, 9, 10, 11, 13, 15, 17, and 18 are shallow wells about 1,300 ft. deep. No. 20, 1,750 ft., 20 gravity oil, came in Mar. 11, 1914, located 3 miles from Nos. 1 to 19 inc.	
...	Naranjos (Lease and fee)	...	Amatlan 20 gravity oil
1...	...	10,000	1,885 ft. deep
2 & 3...	...	...	Dry holes, about 2,300 ft.
4...	...	50,000	Flowed this amount for a week when gate valve broke
5, 6, & 7...	...	...	Locations
...	San Marcos (Lease)	...	South field, 20 gravity
1 & 3...	...	...	Dry at about 2,800 ft.
2...	...	150	Shut in
...	Tierra Blanca (Lease)	...	35 gravity oil
1...	...	...	Abandoned
2...	...	20	Shut in; 3,100 ft. deep
...	Tlacolula (Lease)	...	...
1 & 2...	...	...	Dry holes at 4,100 ft.
3 & 4...	...	...	Drilling and derrick
...	San Pedro (Lease)	...	State San Luis Potosi
1 & 2...	...	10 (each)	Shut in; 45 gravity oil
3 & 4...	...	...	Drilling

Huasteca Petroleum Company (Doheny Interest)

...	Juan Casiano (Fee and lease)	...	Chinampa, 19 gravity
1...	...	400	Shut in; 1,700 ft.
2...	...	100	Shut in; 1,700 ft.
3, 4 & 5...	...		Abandoned
6...	...	10,000	Shut in; 2,070 ft.
7...	...	25,000	Has been flowing 5 yr.; 1,700 ft.
8 & 9...	...	...	Drilling

HUASTECA PETROLEUM COMPANY (DOHENY INTEREST).—(*Continued*)

Well No.	Property	Daily Production	Remarks
	Cerro Azul (fee)		South field
			Reported 27 gravity oil
1			Abandoned
2 to 5 inc.			Shut in
	Juan Felipe (fee)		South field
1			Shut in
		Five wells are reported completed on these two properties, with a combined capacity of 175,000 bbl. daily. Pressures reported from 575 to 1,080 lb. for different wells	
	Pithaya (fee)		Northwest of Casiano
1 & 2			Abandoned
3 & 4			Drilling
	Juarez (sub-lease)		Adjoining Pithaya
1 & 2		Show in No. 1	Shut down
	Ebano field		Fifty miles west of Tampico
1 to 27		8,000	All but 5 shut in; 11 gravity oil

PENN-MEX FUEL COMPANY (SOUTH PENN OIL CO.)

	Alamo Ranch (Lease)		South of Tuxpam River
1		2,000	
2		20,000	
3 & 4			Drilling
	Agua Nacida (Lease)		South of Tuxpam River
1		Show	
2			Abandoned 2,700 ft.
3			Drilling
4			Derrick
	Tamatoca (Lease)		Near Potrero del Llano
1			Drilling at 2,950 ft. last year
	Las Canas (Lease)		South of Tuxpam River
1			Drilling at 3,000 ft.
	Molino (Lease)		
1			Derrick

International Petroleum Company (John Hays Hammond Interests)

Well No.	Property	Daily Production	Remarks
	Tinaja Ranch		South of Amatlan
1 & 2			Dry holes
1	San Marcos		Near Tanguijo, shut down at 3,200 ft.
2			Derrick

Mexican Eastern Oil Company

	San Diego Ranch (Near Dos Bocas)		
No. 1			Abandoned at 2,785 ft.

Electra Oil Company

	Rancho Abajo. (East of Casiano)		
1		Show oil	Drilling

Mexican Premier Oil Company (Spellacy)

	Soledad Ranch. (north of Cuecillos)		
1			Shut down at 2,350 ft.

Hidalgo Petroleum Co. (M. A. Spellacy et al)

	La Calle		
1			Rigging up

Cortez Oil Company (American Tobacco Co. Interests) (Now the Port Lobos Petroleum Co.)

3	Empalizada (Amatlan) (north of Casiano)		Shut down at 2,700 ft.
4		25,000	Shut in. Casing trouble
1	Sandoval		Drilling (offsetting No. 4 Tepetate)

GALEY ET AL

Well No.	Property	Daily Production	Remarks
	San Diego de la Mar		In Chinampa
1			Shut down at 2,100 ft.

STANDARD OIL COMPANY OF MEXICO

	El Gallo Ranch (Lease)		
1			Abandoned at 2,727 ft.

MEXICAN FUEL OIL COMPANY

	Caracol Ranch (11 to 12 gravity oil)		West of Tampico
1			Abandoned, shallow
2, 3 & 7		350	Shut in, shallow
4, 5 & 9			Shut down
8		175	Flowing, 770 ft. deep

TAMPICO FRUIT COMPANY

	Caracol Ranch		Fifteen miles up the Panuco River
1		2	Shut in 2,910 ft.

TAMPICO OIL CO., LTD.

	Chila Ranch		Near Caracol; 12 Gravity
1 & 4		205	Flowing
2 & 3		50	Shut in
	Salinas Ranch		16 gravity oil
2		100	Sanded up; 1,603 ft.
3			Drilling

DOS BANDERAS OIL & GAS CO.

1	Dos Banderas lease (Near Tampico)	Small production	Shut down; 3,480 ft. deep

SMITH OIL CO.

No. 1	Caracol lease		Drilling at 1,925 ft.

Topila Field

Company	Wells, Completed	Wells, Drilling and Locations	Wells, Dry or Abandoned	Estimated Daily Production, Barrels
Mexican Fuel Co. (Pierce Oil Corp.)	4	1	1	750
East Coast Oil Co. (Southern Pacific R.R.)	3	3	1	100
Mexican Oil Co.	1	1	....	2,000
American Fuel Oil Co.	1	....	....	5,000
La Corona (Dutch Shell)	2	1	1	130
Mexican Gulf Oil Co. (Mellon interests)	1	....	1	....
Tampico Panuco Oil fields	1	2	1	....
Topila Petroleum Co. (Spellacy)	1	....	1	....
Penn-Mex Fuel Co. (South Penn)	1	....	1	....
Scottish Mexican Oil Co.	1	....	1	....
M. C. Anderson	2	....	....	52
W. H. Milliken	1	....	....	20
R. L. Brooks (Producers Oil Co.)	1	....	....	450
Cia. Petrolera de Monterey	1	2	....	105
Mexican Associated Oil Co.	1	....	1	....
Cia. Explotadora de Topila	1	1	....	....

Panuco Field

Company	Wells, Completed	Wells, Drilling and Locations	Wells, Dry or Abandoned	Estimated Daily Production, Barrels
Vera Cruz Mexican Oil Syndicate, Ltd	5	1		10,000
International Petroleum Co	4	4	2	800
Transcontinental Petroleum Co	1	1		800
Gulf Coast Corp	3	1		100
Mexican Development Co	1		1	
La Corona Petroleum Co	4	2	1	100,600
Freeport and Mex. Fuel Oil Corporation	5	4	1	31,500
National Oil Co	9	3	1	21,000
Mexican National Oil Co	2	2	1	
Tampascas Oil Co	2	1	1	350
Penn-Mex Fuel Co	1	1		25
Piedras Development Co	1			25
Cia. Petrolera Las Brujas		2		
Producers Oil Co. of Tex	2			6,500
Cia. Petrolifers, S. A		2		
English Oil Co. of N. Y	1	3	1	
Cia. Petroleo Mexicana	1	1		75
Excelsior-Panuco Oil Co	1			1,000
Mexican Gulf Oil Co	4	1	1	12,000
Scottish Mexican Pet. Co	1			250
M. A. Spellacy	1			1,000
East Coast Oil Co	7	3		8,000
Mexican Development Co				
Tal Vez Oil Co	1		1	1,500
Victoria Oil Co		3		
Heradura Oil Co	1		1	
R. E. Brooks		1		
Cia. Petroleo Maritima		1		
La Bonanza Pet. Co	1	1		200
New Bonanza Oil Co		1		
Cia. Abastecedora de Petroleo y Aceite		2		
Yisarri & Blanco		1		
Cia. Petroleo Mexicana La Nacional		1		
Cia. Petroleo Los Perforadores		1		
Cia. Indio de Petroleo		1		
Los Dos Estrellas Oil Co		1		
Cia. Petroleo La Oxaquena		1		
Cia. Petroleo Panuco-Maugaubes		2		
George Harmon		1		
Alamo Oil Co		1		
Cia. Hispania-Mexicana		1		
Panuco-Topila Pet. Co		1		
El Vado Petroleum Co		2		
Panuco Valley Oil Fields Ltd	1			2,000
Pan-American Oil Co		1		
Fenix Oil Co		1		

Summarizing the above, we find that there were in the Mexican fields during the spring of 1915, 53 wells producing, 111 shut in, and 40 abandoned, while the number of wells drilling, shut down or simply located, was 105.

1. *Markets*

All oil exported from Mexico goes by tank ships. During normal times most of the Mexican railroads are equipped for oil burning, and obtain their fuel from the Ebano field, or from the several refineries at Tampico or Miniatitlan after it has been "topped." By far the greater proportion of all Mexican oil which is exported goes to United States ports, where it is "topped" and the residue is sold as fuel or for paving purposes. During 1914, shipments of Mexican oil were as follows:

Destination	Barrels (42 Gal.)
United States	15,476,727
Mexican Coastwise	4,510,061
South America	195,138
Continental Europe	365,205
United Kingdom	67,780
Panama	46,446
Cuba	20,000
Total clearances	20,674,357

Refineries.—There are in Mexico the following refineries which supply the domestic market with refined oils:

	Daily Capacity, Barrels
Standard Oil Co. of New Jersey at Tampico	4,000
Waters-Pierce	Small
Pearson (Mex. Eagle) (Eventually 25,000), Tampico	12,500
Pearson (Mex. Eagle) at Minatitlan	15,000 to 25,000
Huasteca Pet. Co. at Tampico	10,000
Huasteca Pet. Co. at Ebano Asphalt Plant	......

2. *Quality*

The oil produced in these fields is divided roughly as to quality into two classes—one heavy gravity, running from 10 to 14° Bé (0.993 to 0.973 sp. gr.); and one relatively lighter, running from 18 to 22° Bé (0.947 to 0.922 sp. gr.). A small quantity of very light crude oil, of 30 to 40° Bé. (0.840 sp. gr.), has been discovered, but no promising developments of it are assured.

The distillation analysis of this oil yields the following percentage of refined-oil products, compared with Illinois and Oklahoma crude oil:

	Yields of Finished Products from Mexican Crude Gravity 25° Bé Per Cent.	Yields of Finished Products from Illinois and Oklahoma Crude, Process No. 3, Per Cent.
Benzine	7.14	21.00
W. W. Burning Oil	21.42	18.38
Gas oil	21.42	28.81
Neutral Distillate for Red and Miscellaneous Lubricating Oil	11.71	7.68
Asphalt	32.16	17.62
Process Loss	6.15	6.51
	100.00	100.00

And the following analyses of characteristic components and heat value are reported by Messrs. Dow & Smith, Chemical Engineers, New York City:

REPORT OF ANALYSIS NO. 6804, JUNE 11, 1912.

Specific gravity at 60°F.	0.9115
Equivalent to	23.8°Bé
Flash, N. Y. State open tester	77°F.
Flash, N. Y. State closed tester	65°F.
Fire test	120°F.
Boiling point (distillation begins)	175°F.
Water	Trace
Viscosity at 32°F. (Engler 1st 50 c.c.), sec.	1,980
Viscosity at 72°F. (Engler 1st c.c.), sec	298
Loss on heating 20 g, 5 hr. at 325°F., per cent.	26.8
Fixed carbon, per cent.	7.23
Paraffine scale, per cent.	2.25
"Asphalt Contents" 50 pen. (slow evaporation 32 hr. in air bath at 300 to 400°F.), per cent.	59.5
"Asphalt Contents" 50 pen. (quick evaporation by burning), per cent.	52.0
B.t.u.	18,493

Ultimate Analysis

	Per Cent.
Carbon	82.83
Hydrogen	12.19
Oxygen (by difference)	0.43
Nitrogen	1.72
Sulphur	2.83

REPORT OF ANALYSIS, NO. 7288—CRUDE MEXICAN OIL

Specific gravity at 60°F.	0.985
Equivalent to	12.1°Bé
Flash (closed tester)	157°F.
Fire test	225°F.
Sulphur, per cent.	4.20
Scale paraffine, per cent	0.518

Calorific value	18,227 B.t.u.
Viscosity at 60°F	Too thick to take
Viscosity at 100°F	Too thick to take
Viscosity at 212°F. (200 c.c. Engler Viscometer)	615 sec.
Flows readily at	150°F.

Fractional Distillation

	Per Cent.	Products.
1st drop, 140°F.		
140 to 302°F	4	naphtha distillate
302 to 545°F	20	burning oil distillate
545 to 600°F	10	cracking commenced
600 to 620°F	15	cracking continued
Residue and loss	50	cracking badly, temperature fell to 605°F.
Penetration at 77°F. of residue from fractional distillation	60	

Asphalt Contents by Evaporation

100 penetration at 77°F., per cent	70.1
Ductility of ditto, cm	100.0
50 penetration at 77°F., per cent	65.7
Ductility of ditto, cm	100.0

The following are analyses of the Huasteca oil:

HUASTECA OIL,

CASIANO FIELD, WELL NO. 7,

SAMPLE I

Gravity 19.4° Bé	Pour, 0
Flash, 80°F	B. S., 3 per cent.
Viscosity, 450 B. O. T	B.t.u. per pound, 19,124

Fractional Distillation

Temperature	Per Cent.	Gravity, Degrees Baumé
Over from 259 to 385°F	10	58.7
Over from 385 to 493°F	10	....
Over from 493 to 500°F	1	45.9
Total		
Distillates	21	51.5
Residue	78	13.5; pour, 40
Loss	1	
	100	

SAMPLE II

Gravity 19.3° Bé	Pour, 0
Flash, 96	B. S., 2 per cent.
Viscosity, 500 B. O. T	B.t.u., per pound, 19,160

Fractional Distillation

Temperature	Per Cent.	Gravity, Degrees Baum
Over from 244 to 400°F	10.0	57.8
Over from 400 to 493°F	10.0	....
Over from 493 to 500°F	0.6	45.1
Total:		
Distillates	20.6	50.8
Residue	78.5	13.4
Loss	0.9	
	100.00	

3. *Costs and Prices*

The cost of production of oil (of about 12° Bé.) at Panuco at present is about as follows:

Cost per Barrel of 42 Gal.	U. S. Gold, Cents
Bar dues (*50c. Mex. per Metric ton or 6.6 bbl. ± av.)	3.8
Production tax (30c. U. S. per Metric ton)	4.5
One-tenth Royalty (based on an arbitrary value of 50c. U. S. per barrel)	5.0
Transportation by barge, Panuco to Tampico	6.0
Lighterage (terminal to shipside at Tampico)	2.0
Production expenses (estimated)	10.0
Cost of barrel of oil, f.o.b. ship at Tampico	31.3

(*Note: This is Mexican gold with an exchange value of 2 :1).

The cost of transportation by tank ship to United States ports may be estimated as 10c. per ton per day. As the voyage consumes from 3 to 10 days, depending upon conditions and the port to which the oil is consigned, the total cost to destination will vary from 35.5 to 45.3c. per barrel. Costs from Tuxpam will differ with differing producing conditions in the field; pipe line instead of barge transportation, and no bar dues; but with a slightly longer sea voyage for tankers.

The official market price during 1914 for Panuco heavy oil f.o.b. ship at Tampico, ranged from 40c. a barrel in January to 20c. (U. S. gold) at the close of the year. Evidently some oil was sold at less than the cost of production, in order to keep equipment in use, and to take care of production which could not be shut in. Very little oil is sold at the wells in the Mexican fields, although a few sales have been reported at Topila and Panuco.

The lighter oils (19 to 27° Bé.) from the southern districts, are all sold under contract at higher prices than the Panuco oil. Casiano oil is valued at from 50 to 60c. f.o.b. ships at Tampico, which would make its value at United States ports from 55 to 75c. per barrel. The oil from the properties of the same company at Jan Felipe and Cerro Azul is reported to be as light as 27° Bé. The oil from the Alamo lease of the Penn-Mex Fuel Company is about 24° Bé., which is slightly higher than that from the Potrero del Lano lease of the Mexican Eagle Petroleum Co.

4. *Drilling*

Drilling is carried on in these fields with heavy combination rigs, usually of the California type. The rotary is used through the soft upper marls (Mendez or Los Esteros beds) into the upper portion of the San Felipe formation. Here the casing is set on top of some hard limestone shell, and drilling is continued with percussion tools through the San Felipe and well into the massive Tamasopa lime, unless oil is found sooner. In most of the large wells the oil flow has been struck in broken blue shale or limestone, a short distance above the top of the Tamasopa. In others, the oil is found in the upper few feet of the Tamasopa. There are no defined oil "pays" which can be correlated with each other from one field to another, or even from one well to another in the same field. The oil occurs in fractured zones, in which solution channels caused by circulating underground waters are frequent. This condition has led to the generalization, based on the experience of operators in these fields, that all the big wells get their oil "high up;" or in other words at a shallower depth than the average for the locality.

Drilling is very expensive in these fields. An operator expecting to drill must estimate that his first wells, if wild-cats, will cost in the neighborhood of $65,000 each, exclusive of the leasing expenses. This excess outlay may be divided under the heads of high wages; the establishment of camp and commissary; the transportation and duties on drilling materials; the delays due to accidents and waiting for materials, or the cost of keeping a large stock of extra parts to prevent such delays; road building and rights-of-way for roads and water lines; the high cost of fuel, for wildcat wells; and the cost of keeping up communication with Tampico, by means of launches and horses.

The difficulties and expenses of drilling are affected by geological conditions also. Along the coast line, and the shores of Lake Tamiahua, there are deposits of varying thickness, and of Quarternary and Recent age. This is also true in the Panuco field, where they are about 100 ft. thick, and obscure all evidence of structure and intrusion. The Mendez beds vary considerably in thickness within the same district; and the same appears to be true of the San Felipe series. There are no horizons within either of these formations which can be correlated, even between two nearby wells. And owing to the difficulty in determining exactly from a well record the top of the Tamasopa limestone, due to the gradual transition from the San Felipe series to a massive limestone phase near the bottom, it is impossible to ascertain positively the extent and position of such unconformities as probably exist. This is even more difficult because of the lack of cooperation upon the part of the various companies doing geological work in these fields. The isolated location of the various districts permits a degree of secrecy possible in few other fields; and several of the large companies have carried this policy to an unjustifiable extreme. This does not apply, however, to the Panuco and Topila districts.

The entire oil-producing area is crossed in various directions by igneous dikes and intrusions. These are generally in the form of basalts

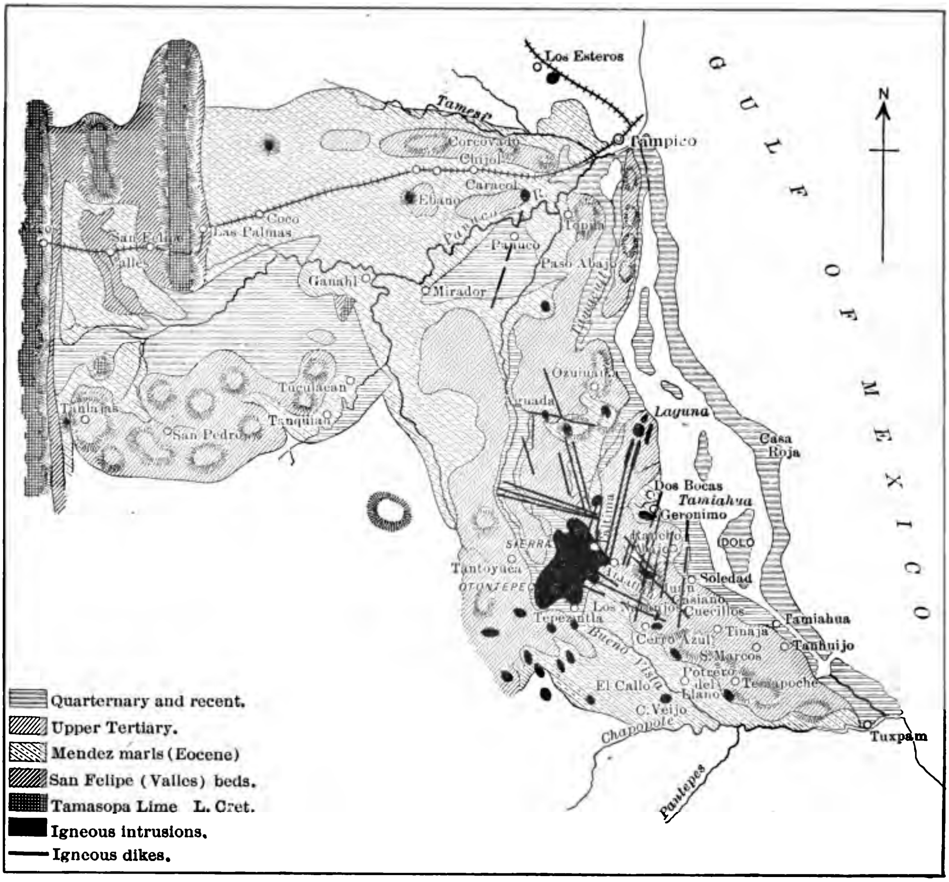


FIG. 2.—GENERALIZED SKETCH OF THE MEXICAN OIL FIELDS, SHOWING AREAL GEOLOGY, LOCATION OF MAIN BASALTIC INTRUSIONS, AND STRIKE OF THE MAIN DIKES IN THE CENTRAL DISTRICT.

and diorites, and at various places occur as very large intrusions. The largest of these is the Sierra Otontepec, shown in Fig. 2.

As there are no common carrier pipe lines, and all companies owning their own lines have enough oil of their own to fill contracts, there is no sale for oil in the field. Therefore the small operator sometimes gets a white elephant on his hands by drilling a good well in a wild-cat district. He may sell outright at considerably less than the well is worth, if he can find a purchaser; or promote a company of his own to handle the well; or enter into an alliance with others who will furnish money to build pipe lines to seaboard, erect a terminal, charter or build tank ships, and finally market the oil in the United States or elsewhere. As matters stand at present, with so many wells shut in, the finding of the oil is the least important step, particularly if there is not enough territory leased in the vicinity to protect such a well, and justify capitalists in undertaking the further development of the project.

IV. GEOLOGY

1. *Stratigraphy*

In the Mexican fields four distinct formations are encountered: (1) an upper series of fossiliferous Tertiary sandy limes and sandstones, interbedded with limy and sandy clays, the beds varying in thickness from 600 to 1,300 ft.; (2) an intermediate section, 2,000 to 3,500 ft. thick, of gray marls and shales (called Mendez marls, or Los Esteros beds), the upper portion of which has been indentified as Eocene Tertiary which give place in their lower portion to (3) the San Felipe or Valles beds of limestone shells alternating with blue and brown shales 200 to 700 ft. thick. These lie upon (4) a massive blue-gray (Tamasopa) limestone formation, at least 3,000 ft. thick, fossilliferous in its upper portion, of Lower Cretaceous age. In some areas, the upper Tertiary limestones have been eroded away, and the surface formation consists of marls (see Fig. 2).

a. The Upper Tertiary.—The upper Tertiary formations have been divided by some geologists into the western or Tanlajas series, and the eastern or Ozuluama and Temapache series. While they keep their general character as marine deposits of rapidly alternating sandy limestones and clays, and local beds of fossils such as the beds of nummulites and orbitoides in the central districts can be correlated for short distances, yet such correlations cannot be carried from one area into another without much care.

In the western or Tanlajas district these beds average about 1,100 ft. in thickness, while in the easternmost wells—those at Tanguijo—they are at least 1,200 ft. thick. These upper Tertiaries, however, practically cover the southern and central portions of the Mexican field, except where river valleys have exposed the underlying marls by erosion, or where large basaltic intrusions have taken their place.

These upper Tertiaries have been found oil-bearing at one locality only—Tanguijo. Here the Aguila company has drilled a number of wells, which are capable of pumping about 500 bbl. per day. The oil is 21°Bé. and the wells decline rapidly. Near the bottom of the upper Tertiary section in the Tuxpam district, a dark bituminous sand bed is reported. This does not appear in the northern section. However, owing to the friability of these beds, and the prevalence of strong jointing, one finds a great many asphaltic seepages exuding from them at the surface. In most cases this is due to the fracturing of the underlying beds, and the intrusion of dikes; the oil finding an easy egress through these upper beds. It is believed by some that the oil found in these beds at Tanguijo has found its way there through migration from lower formations, through fractures. I regard this as very probable. The size of these wells and their behavior hardly encourages prospecting in these beds for the development of commercial production, especially while wells in other districts are so prolific.

While there are numerous exposures of these upper Tertiaries in the central and southern districts, and good clinometer readings can be obtained at many points, yet such data frequently prove misleading. The absence of good key horizons makes it difficult to locate fault lines, or to determine their throw. It is also frequently impossible to work out the details of the monocline structure found in these beds; but since these minor irregularities are probably of little importance, this difficulty is not of great consequence. In other words, intrusions and fractures, channelling in the oil-bearing limestone, and the effect of sedimentary gradients and unconformities are all of so much greater importance in these fields in their influence on oil accumulation, that only the large well-marked structural features need be considered, where these other factors are not known to exist.

b. The Mendez Marls.—The Mendez marls consist of a very uniform deposit of gray to blue shales and marls. In regions of steep folding, these often show bold jointing near the surface. There is practically no change in their lithological character from top to bottom. They average from 2,000 to 3,500 ft. in thickness. In the Huasteca country, they are nearer the former thickness. A few irregular beds of sandy limestone are reported in this formation, but they are not persistent. However, there are frequently small quantities of heavy oil immediately below them, although never enough to sustain production of commercial importance.

The apparent thickening or thinning of these beds within short distances is sometimes accounted for by faulting of the formations. Or a local steepening of the dip may cause an apparent thickening. The upper limestone lenses in the San Felipe beds likewise come in and go out, and seem to indicate unconformity; so that the thickening of the Mendez marl beds is counterbalanced by a smaller thickness of the lower San Felipe series, the depth to the top of the Tamasopa limestone remaining practically constant. Even this interval, however, is not always constant in the same district, for the reason that there is more or less unconformity between the top of the Tamasopa and the overlying San Felipe shells.

c. The San Felipe Formation.—This may be described as a transition series between the upper Mendez marls and shales and the underlying massive Tamasopa limestone. It begins with an occasional thin limestone shell. These increase with depth in number and thickness, being interbedded with blue shales which conversely decrease in thickness downward until the series gives place to massive limestone. These beds apparently vary in thickness from about 300 to as much as 800 ft. Since drillers frequently do not begin to record these shells until they are about ready to set the casing, it is impossible to estimate the thickness of the series in every well merely by the well log.

These San Felipe beds are frequently the oil-bearing formation in the Mexican fields, particularly for the largest wells. This is because of the well-known occurrence of strong fracturing, with which large oil accumulations are associated. Oil is found in such a fracture as high as it can go; and any well drilling into such a "fissure" will encounter its big flow "high up," in the terms of the operator; the marls finally proving an effective seal against its escape above a reservoir in the San Felipe. Among others, the following "gushers" never reached the Tamasopa lime: Juan Casiano No. 6 and No. 7; Pearson's Los Naranjos No. 1 and No. 4; La Corona No. 5 at Panuco, and Spellacy No. 1 at Panuco. In such cases, the oil is usually found in broken shale or channelled limestone in the San Felipe formation. However, since this channelling is probably more frequent at the base of the series, where there is considerable unconformity, the top of the Tamasopa lime has come to be looked upon as the "pay" formation.

While such faulting or fracturing extends to the surface through the Mendez beds, as indicated by dikes and seepages, yet it is perfectly evident that as these upper formations contain no porous beds, no oil accumulations can be looked for above the harder San Felipe. As these soft marls have undoubtedly been pinched together after disturbance took place, they have in many instances effectually sealed any outlet except such passageways as those kept open by igneous intrusions. However, such seepages may travel laterally for considerable distances along joint cracks in the harder upper Tertiaries, before finding an outlet to the surface—making the successful location of wells problematic if based upon no other evidence than a seepage of *chapopote.*

d. The Tamasopa Limestone.—Jeffreys says that the limestones of the San Felipe series can be distinguished from the lower Tamasopa lime in that (1) they contain no fossils; (2) they are not massive; and (3) their texture is generally of more even grain than that of the Tamasopa. These differences, however, are not always easy to detect in a well sample. The Tamasopa is a compact, hard, gray, crystalline limestone, fossiliferous near the top. It has never been drilled through, and no productive horizon is known far below the top. There are no oil seepages or evidences of hydrocarbon contents at the outcrop, far from the contact of overlying formations. There appears as yet no conclusive evidence that the oil in the Mexican fields had its origin in the beds of the massive Tamasopa limestone, as claimed by some geologists. The very nature of such beds, especially considering the subsequent period of erosion, is against such an assumption. On the contrary, there is more evidence here than in many other fields pointing to the probability of its having migrated from the overlying marine marls. This will be discussed in another paragraph.

2. *Structure*

All the large wells drilled up to the present are located where there exists a significant combination of both favorable anticlinal or dome structure with pronounced fracturing of the formations. These fractures (frequently faults of relatively small throw) are usually accompanied by basaltic intrusions, and seepages of asphalt and gas. In the Panuco field, all surface evidence both of structure and intrusion, has been obscured (except at one point north of the river in the Tampalache area) by about 100 ft. of alluvial sediments; but drilling has shown that the same conditions exist and have influenced the oil accumulation here as in other parts of the Mexican fields (see Figs. 3 and 4).

There has also been faulting in connection with some of these anticlinal structures, especially those nearer the mountains (Fig. 5). These folds become broader and less frequent to the eastward (see Fig. 5). However, other forces caused a number of relatively well-marked folds in the vicinity of Otontepec, such as those at Potrero del Llano and Los Naranjos. This folding was caused by lateral thrust and probably certain vertical stresses incidental to the formation of the Sierra Madre mountains to the west. These in turn set up other lines of weakness in the formations, through which during late Tertiary time other igneous rocks were intruded. This is shown by Fig. 6, which is a map of the central part of the fields in which many of the main basalt dikes have been located by the author.

A study of this map will reveal a number of interesting relations, for instance, the general agreement between the strike of the sedimentary formations and that of the main dikes in the coastal portion of the fields. A reference to the sketch map (Fig. 2) showing the general areal geology of the Mexican field, fails to reveal any locality where the Tamasopa lime or even the San Felipe beds have been thrust up to the surface by intrusives, as claimed by some of the earlier writers. The writer knows of no instance where any pronounced doming has been caused by the upthrust of dikes or so-called "plugs" of basalt. Some very local distortion and faulting has been caused at certain places; but such instances are balanced by others in which the sedimentaries actually dip toward

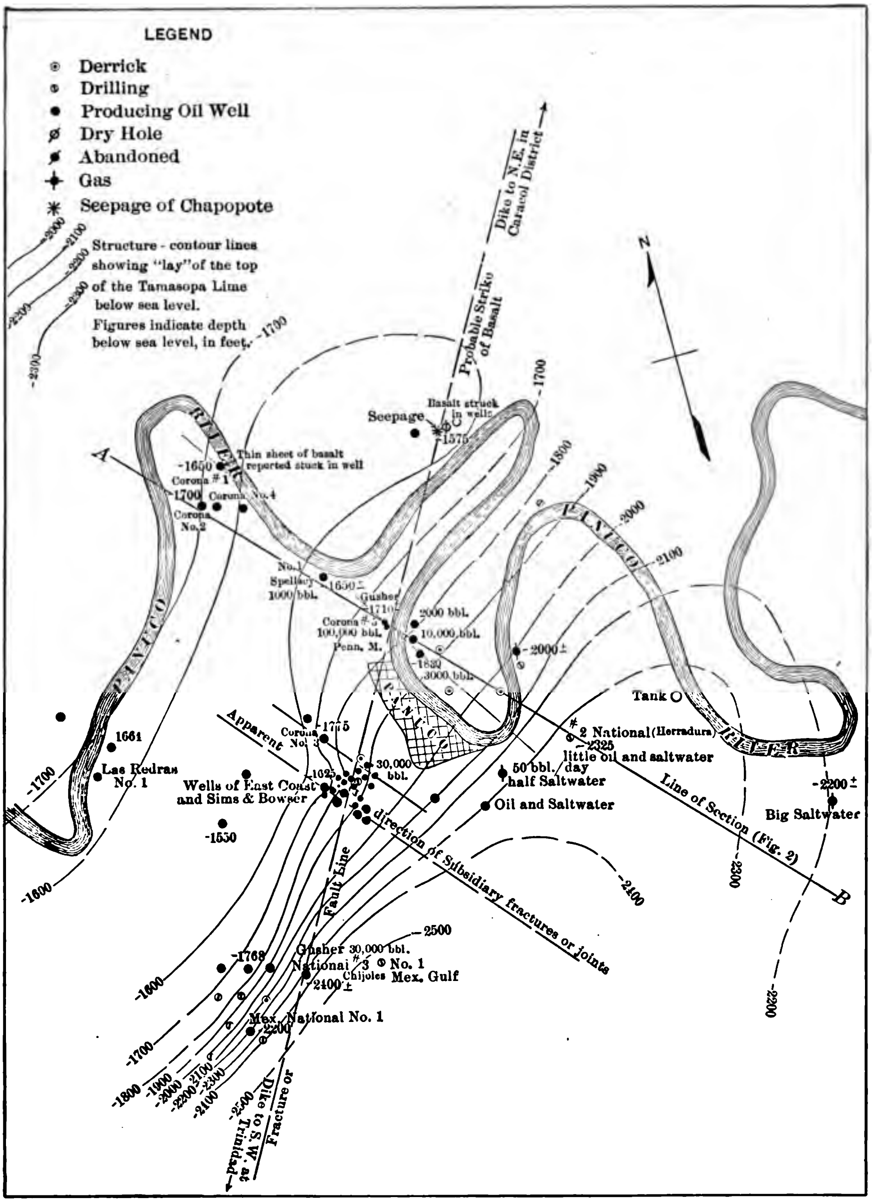


FIG. 3.—MAP OF THE PANUCO OIL POOL, SHOWING "LAY" OF THE PROBABLE TOP OF THE TAMASOPA LIME, AS INDICATED BY WELL LOGS; AND ALSO THE LOCATION OF PRINCIPAL WELLS.

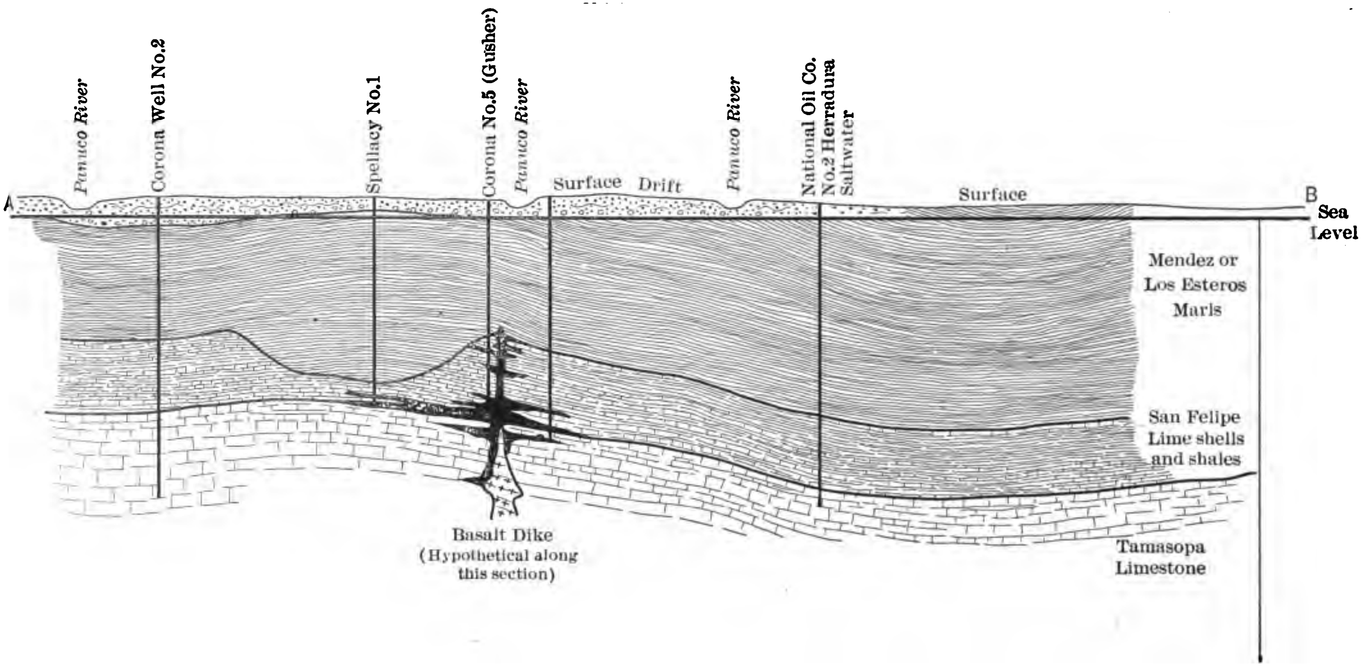


FIG. 4.—HYPOTHETICAL SECTION THROUGH THE PANUCO FIELD, MEXICO, ALONG THE LINE A-B (FIG. 3), SHOWING ANTICLINAL TERRACE AND FRACTURE.

(Note: The character of the unconformity as shown between the marls and the San Felipe beds is tentative.)

large igneous bodies from all sides. That is to say, shrinkage due to cooling has more than offset any "doming" effect.

In common with other fields where the surface beds are friable, the local dip of such beds as measured by a clinometer is apt to be much greater (especially near intrusives) than that of the underlying formations. Additional drilling has failed to corroborate the "nail" theory of the effect of igneous plugs or other intrusions upon the structure of surrounding sedimentaries, in the Mexican field.

A reference to Fig. 7 will show that some secondary folding by lateral thrust has been caused by the intrusion of the Sierra Otontepec—the largest igneous mass in the field west of the mountains.

Igneous Intrusions.—Again referring to Fig. 6, it will be seen that the fields at Juan Casiano, Los Naranjos, Dos Bocas and Panuco are all located at the intersection of strong fractures, where such intersections occur on anticlinal folds. Access to the well records of certain operating companies would undoubtedly reveal interesting data with regard to the relation of the accumulation of oil under these conditions to possible fault displacement. Where the marls are exposed at the surface, as at Los Naranjos and Dos Bocas, it is impossible to determine local fruits from surface indications, unless they have considerable throw or linear extent, enough to make it possible to locate them by inference. Nor is it at all easy to determine the throw of such possible faults, even where the harder Tertiary beds are found at the surface, due to the lack of well-defined key horizons. However, sufficiently detailed work will usually give enough clues to their existence, especially after a knowledge of conditions in the surrounding areas has been gained.

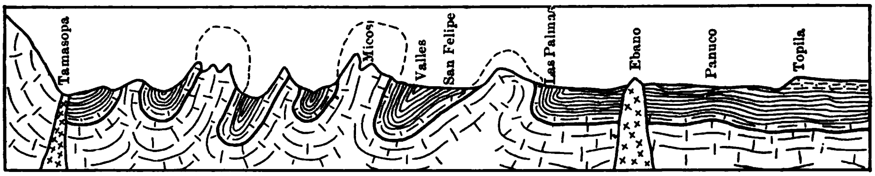


Fig. 5.—Generalized Section East and West through Northern Part of the Oil Fields. After Jeffreys.

Intersections of strong fractures are frequently accompanied and marked at the surface by conical basalt peaks, which usually represent the "mushrooming" of an igneous neck intrusion. Wells drilled close to the contact at several of these conical hills have disproved the theory, advanced by one geologist, that they were "plugs" whose conical shape persisted at depth. Such wells have frequently started close to the contact and have been drilled into the oil formation at more than 2,000 ft. in depth, without encountering any further basalt, or any violent distortion. These intrusions and accompanying flows apparently follow some fracture intersection where the resistance was least, and merely represent knots along the line of a dike (Fig. 6). At other places cone-shaped hills occur, also along the line of some fracture or dike-line projected, but with no sign of basalt at the surface, and no evidence of violent folding. The formations at such places seem to be in place, yet considerably harder than the surrounding district. It is quite probable that some

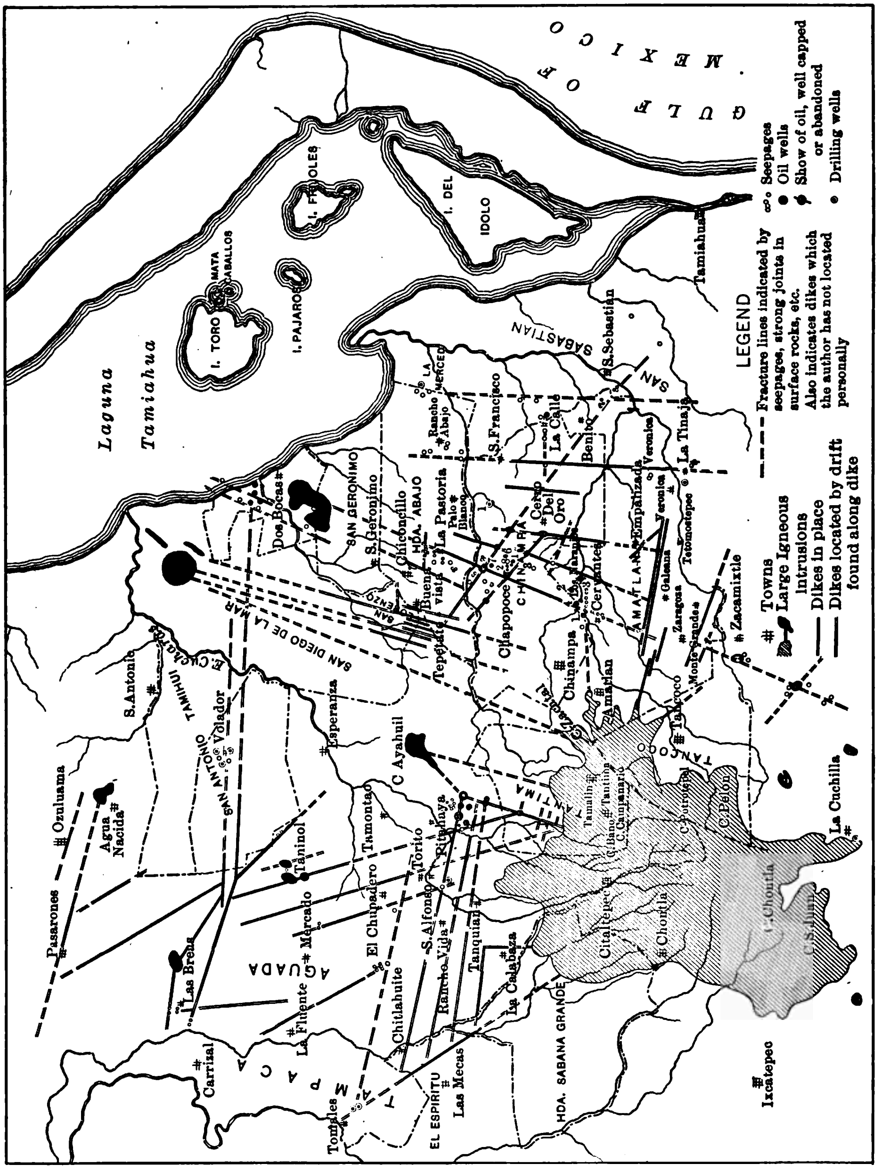


FIG. 6.—MAP OF A PORTION OF THE MEXICAN OIL FIELD, SHOWING THE RELATION OF THE PRINCIPAL IGNEOUS INTRUSIONS TO OIL ACCUMULATION AS EVIDENCED BY SEEPAGES AND LARGE GUSHERS WHICH HAVE BEEN DRILLED AT DOS BOCAS, JUAN CASIANO, AND LOS NARANJOS.

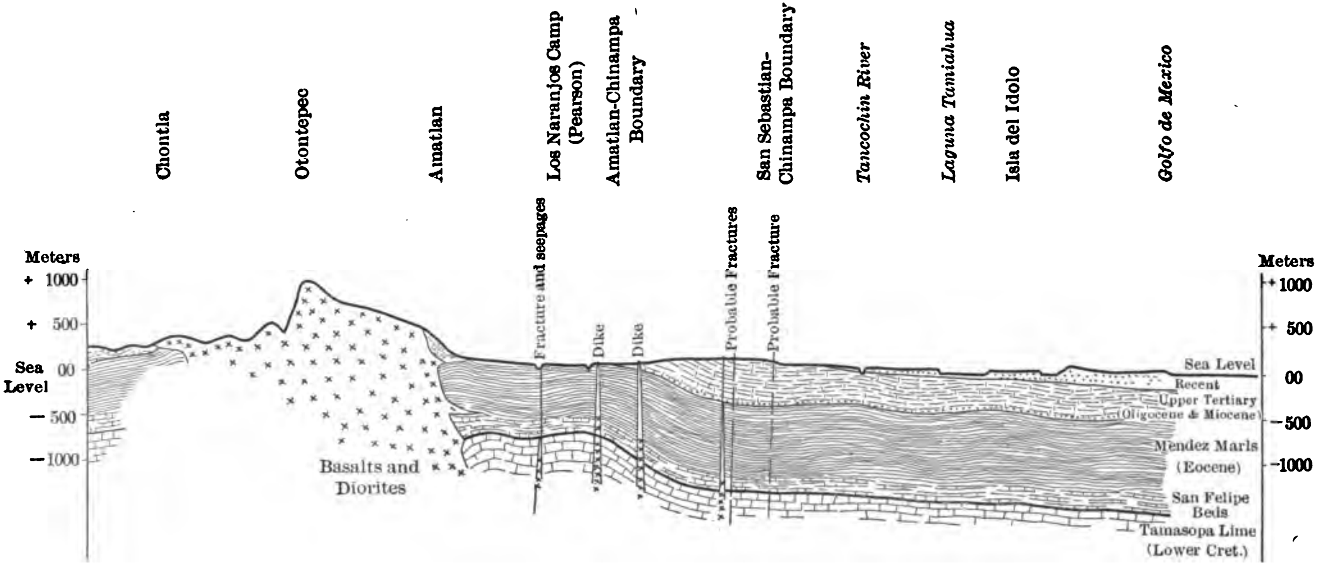


FIG. 7.—DIAGRAMMATIC GEOLOGICAL SECTION ACROSS LOWER PART OF FIG. 6.

such intersections, not filled with basalt to the surface, offered a channel for the circulation of underground water more or less hot or highly mineralized, which metamorphosed the sedimentary formations in the immediate vicinity. These formations would thus resist erosion much more readily than the softer limes and sands about the channel. One notable example of this is Cerro de Zaragosa, between Amatlan and Zacamixtle This has every appearance of being a typical basalt peak, yet examination failed to show any basalt on its sides, which are composed of upper Tertiary formations. And yet the peak is directly in line with a main series of dikes extending from near Dos Bocas to Zacamixtle, through Juan Casiano and Los Naranjos. (Fig. 6.)

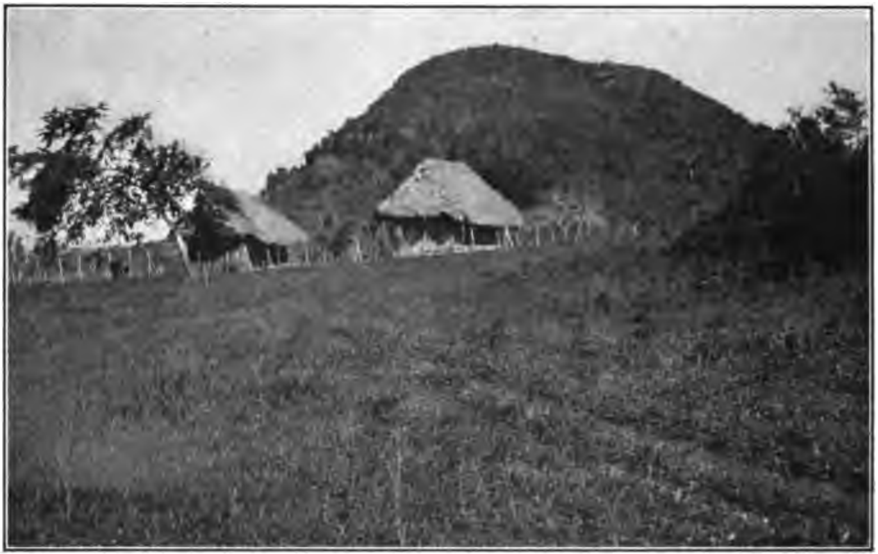


Fig. 8.—Cerro Taninul, a Conical Basalt Peak Typical of Those Occurring at Many Points in the Mexican Oil Field.

Another example is the eastern peak of the two on the edge of Aguada, called Taninul. The western one is composed of basalt (Fig. 8), while the smaller eastern peak shows only large blocks of a metamorphosed marl containing numerous small siliceous nodules. This peculiar rock is found at a number of places in the vicinity of large intrusions of basalt, and has been caused by the circulation of underground currents, probably both hot and more or less mineralized, as pyrite is frequently found nearby. There are also numerous smaller peaks and conical knolls throughout the field.

3. *Source of the Oil*

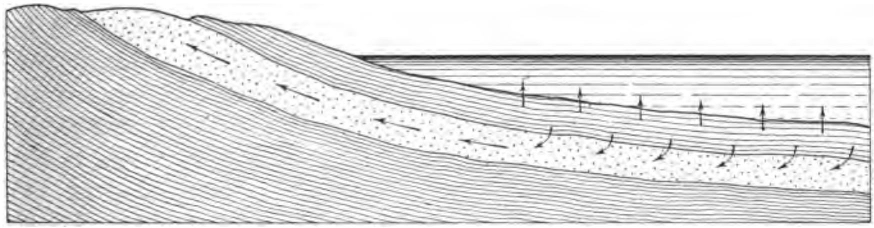
In a consideration of the source of the oil found in the wells in the Mexican fields, the following facts hold true at all points:

1. The oil is found in a porous and usually fractured limestone (sometimes shale) near the top of the Tamasopa limestone formation. Although a few shows, and, in some cases, large quantities of salt water, have been encountered, no oil in large quantities has ever been found as yet by drilling deeper into the lime.

2. Oil is also found, particularly in the "gusher" wells, in the broken lime shells and blue shales of the San Felipe series, usually under conditions indicating strong fracturing and jointing.

3. Oil is not found in quantity in the homogeneous marls overlying the San Felipe, although these marls are more or less petroliferous throughout, as shown by drilling. However, in drilling near dikes and fractures when seepages occur at the surface, shows of gas and heavy oil are often encountered in the hole all the way down.

4. These marls were laid down upon already compacted limestone beds of great thickness (Tamasopa) in which, so far as now proved, no porous horizon existed. It is not considered probable that oil could have been formed within massive beds of marine limestone such as these.



After King, 19th Annual Report, U. S. Geological Survey, part 2, p. 80.

FIG. 9.—DIAGRAMMATIC SECTION SHOWING THE FLOW OF CONNATE WATER, OIL, AND GAS, DUE TO CONSOLIDATION OF SEDIMENTS.

5. Any oil coming from below the limestone must have been liberated by fracturing and intrusion and subsequently caught and held by the denser marls, pinching together and retaining such oil in the more easily shattered and hence more porous San Felipe and upper Tamasopa lime. Circulation of underground waters along the unconformity on top of the Tamasopa may have aided in forming reservoirs for the oil. There is considerable evidence of these channels in some wells, but they may have been formed after fracturing had promoted circulation.

This (5) presupposes either the inorganic origin of oil, or the existence of enormous previous accumulations of it in formations at great depth—below the Tamasopa. The latter alternative does not seem reasonable, because such previous accumulations would already have been more or less localized in certain areas, and when allowed to filter upwards through say 10,000 ft. of limestone after later fracturing, would have shown much more varied composition, and would have been restricted to more local and erratic areas in the upper formations in the Mexican fields, instead of being so generally distributed under so nearly the same conditions.

The author holds it much more reasonable to assume that the oil had its origin in the petroliferous marine marls of the Mendez formation. The compacting of these marls under a constantly increasing overburden, as demonstrated by King (Fig. 9), would have forced the liquid content into any more porous bed existing below. It must be remembered that such marls are not impervious to the movement of fluids at the time of compacting, although they gradually become relatively so. As there is evidence of much channelling by hot water along the top of the Tamasopa, these channels offered both a course for the necessary movement, and the porosity for concentrating such oil as found its way into them. As stated by Roswell H. Johnson, "If beds of less compressible material (porous) meet or underlie the shales in question, there will frequently be a lateral movement along such beds to some point where upward movement will be resumed." There may be even today such compacting and lateral movement still going on along the shores of the Gulf of Mexico, which would be most actively manifested along fracture and fault lines where circulation upward could be resumed most freely. The greater the opportunity for this movement, while at the same time there was a chance for the segregation and retention of the oil, the larger and more extensive would be the final accumulations. Later movement along bedding planes of the marls probably found ideal catchment areas in fractures which had been formed in the meantime, and added to the accumulation of oil already there. This may account for the different composition of oils in the different districts.

Many phenomena noticed in connection with the production of oil in these fields are worthy of study. Employees of the Mexican Eagle Petroleum Co. report that when sudden barometric changes occur, such as those accompanying the frequent "northers" on the Gulf coast, the closed pressure of the Potrero del Llano well No. 4 increased as much as 100 lb., while nearby wells also show an increased pressure of smaller extent. If true, whether this indicates a connection between the underground reservoir and the Gulf through channels or fractures in the hard lower beds, or (what is more likely) direct connection of the oil and water table to the surface through fractures kept open by intrusives, is a question upon which more data are needed.

a. Basalt.—A reference to Fig. 2 will show that while the master intrusion in the central part of the Mexican fields is the Sierra Otontepec, and that radiating and tangential dikes in the immediate vicinity may be due to secondary fracturing caused by this large intrusion and its subsequent shrinkage; yet in general this represents only an exceptionally large example of a type common throughout the field.

So far as worked out by the author (Fig. 6) the long fracture lines (usually filled by basalt dikes) were the primary phenomena. These are intersected at various angles by shorter fractures which resemble strong joint cracks at the surface, and which are less frequently filled by igneous intrusions. These are sometimes filled by sandstone dikes, as in the vicinity of Juan Casiano, and are in places indicated by short series of asphalt seepages following a general direction. As at such intersections there are so frequently found conical basalt peaks, or even hills of metamorphosed sediments, one soon comes to the conclusion that these knobs were merely phenomena incidental to and accompanying the main dikes. There is a group of such hills at La Pithaya and Juan Casiano.

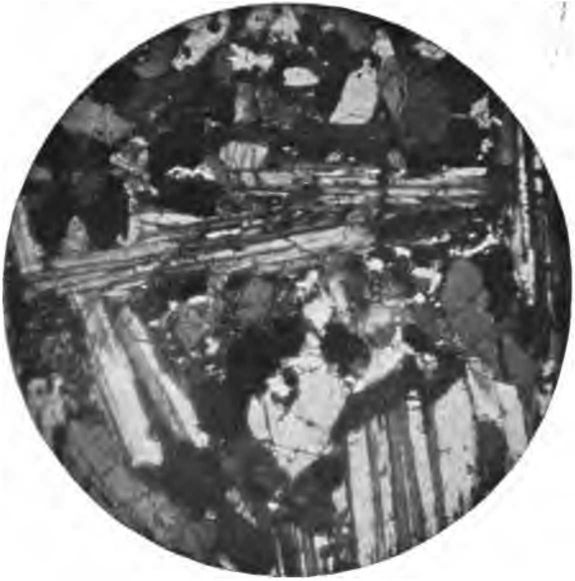


FIG. 10.—OLIVINE-DOLERITE. MAGNIFICATION ABOUT 22 DIAMETERS.
Augite, olivine, and magnetite in well-formed crystals, and evidently crystallized before the feldspar. Labradorite in hemimorphic lath-shaped crystals surrounding the other constituents.

In neither case has the size of any one of these peaks, nor the distortion of the sedimentaries accompanying it, been enough to cause fractures of such length as are known to extend from these localities both north and south. The immediate presumption after studying these localities is that such fractures are the main fact to be taken into consideration.

Now, it will be noticed in Fig. 6 that Otontepec lies at the intersection of the two main series of dikes found in the central district—one extending approximately 20° W. of N. toward Panuco, and the other at about the same angle east of north toward Dos Bocas and the Gulf. These fractures are so persistent that the same conclusion appears to follow as was stated in the preceding paragraph, viz., that Otontepec is merely

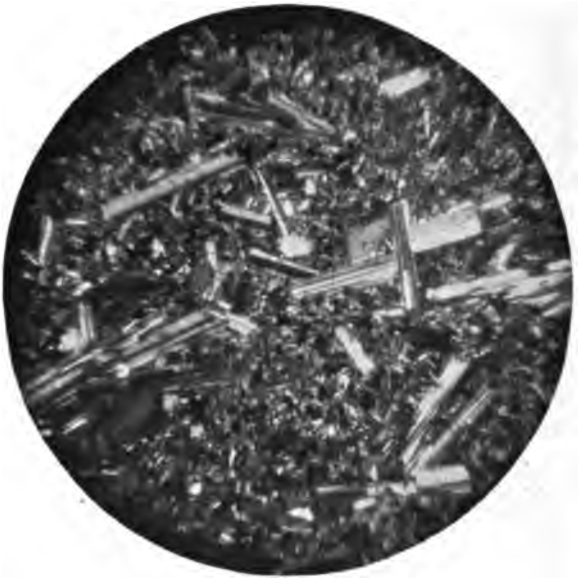


Fig. 11.—Specimen from a Small Dike. Magnification 40 Diameters.
Larger proportion of augite than Figs. 10 or 12.

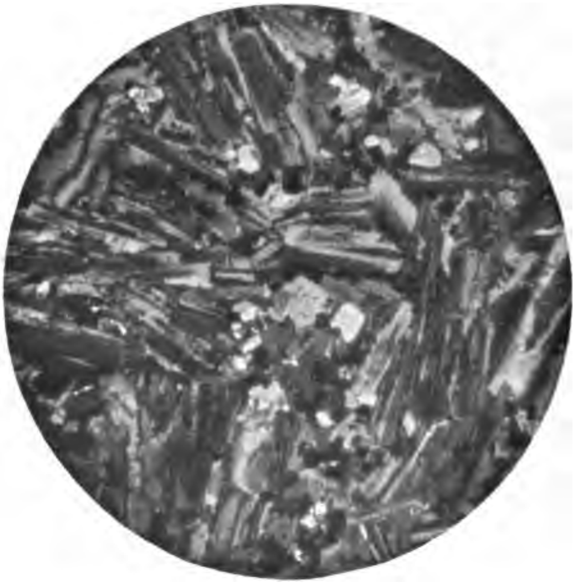


Fig. 12.—Olivine-Dolerite. Magnification 22 Diameters. Same as Fig. 10.

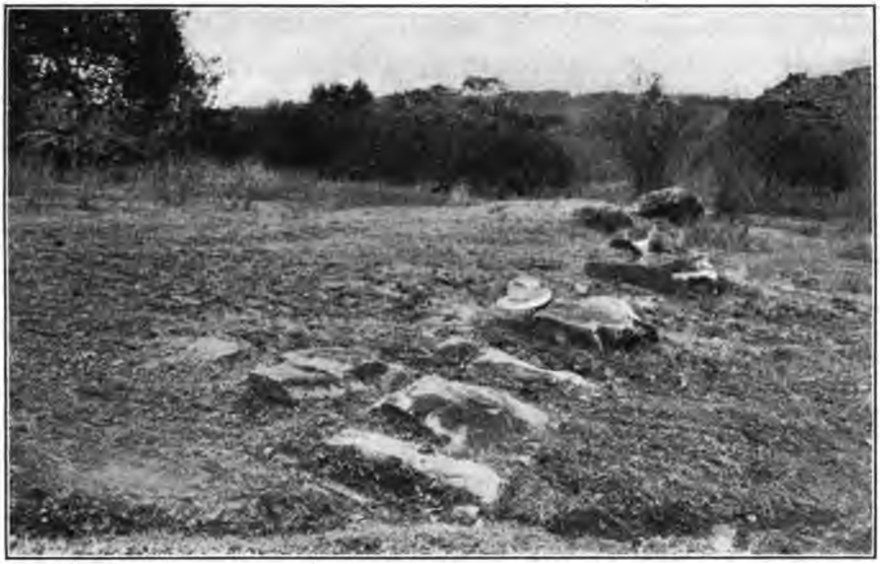


FIG. 13.—BASALT DIKE APPEARING AT THE SURFACE IN THE MEXICAN FIELDS, NEAR OZULUAMA.

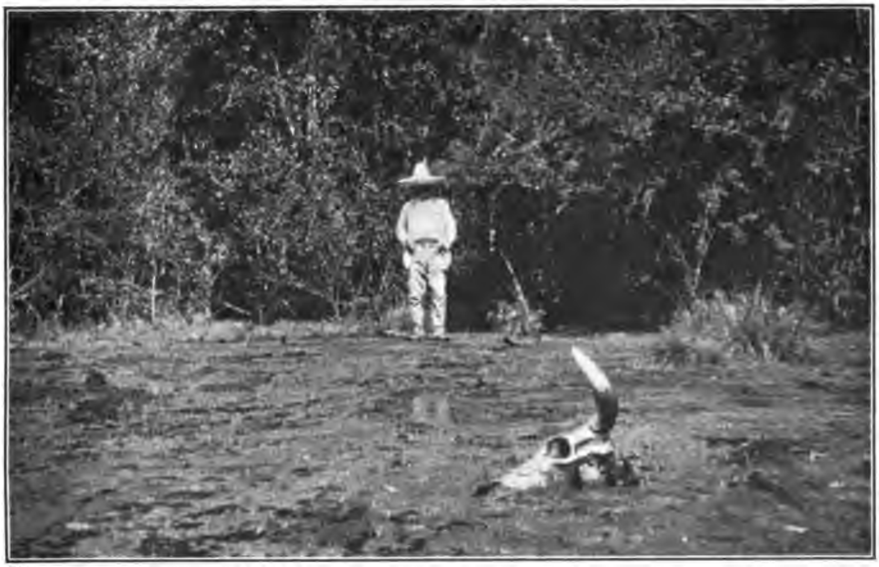


FIG. 14.—PORTION OF A LARGE SEEPAGE OF "CHAPOPOTE" IN THE CENTRAL DISTRICT OF THE MEXICAN OIL FIELDS.

Notice the remains of the cow which became entrapped in the bed of asphalt.

an intrusion which represented the point of greatest weakness in the sedimentary rocks. It probably was formed by the coalescing of a number of necks which found their way to the surface as did the smaller ones found at other points. A study of the districts south and west of Otontepec would probably yield confirmatory evidence. As it is, there are no published data on these districts.

The accompanying sections (Figs. 10, 11, 12) were taken from specimens of basalt dikes in this central district.

b. Seepages.—Seepages of asphalt, sometimes accompanied by gas and occasionally by sulphurous brine, are found throughout the Mexican field, and, with the relatively horizontal strata and igneous intrusions, constitute a unique feature of this oil field. Several of the largest groups of seepages (Fig. 6) are found on anticlines, and most of them are near dikes or intrusions which reach the surface. Others, however, appear to exude from the surface formations where there is apparently no structural reason for oil accumulation below. However, experience and inference have led to the general belief on the part of geologists that in most instances seepages indicate an intrusion of igneous rock in the vicinity not far from the surface (Fig. 14).

It is common to find short lines of seepages intersecting a main dike at a high angle (Fig. 6). A series of such short lines of seepages, each at an angle to the strike of that district, yet taken together following the strike of a dike, is often evidence by which a fracture can be projected from another locality. The shorter intersecting fractures probably in most cases represent joint cracks due to local pressure. The photograph (Fig. 13) shows a spur of basalt from the dike following such a direction.

Seepages are also found exuding from basalt high on the Sierra Otontepec, at a considerable distance from the contact of the sedimentaries. As there are many open channels through the basalt which would obviously offer less resistance to the passage of oil than would the overlying Mendez marls, these apparently anomalous occurrences are not so in reality. Basalt dikes cannot be looked upon as barriers to the movement of oil, except so far as they accompany fractures which tend to entrap it. Such migration may be from several directions.

A study of the central district shows a significant resemblance between the fractures located and a series of joint or shrinkage cracks which might be due to a compacting and drying of the sedimentaries. A remarkable persistence of roughly triangular areas inclosed by these fractures, and the fact that their persistence in certain general directions has some relation to the direction of the main stresses at the time of the Sierra Madre uplift, suggest that such general jointing or shrinkage cracks may have been the primary cause of the lines of weakness which were followed later by intrusions.

4. *Limits to Prospective Fields*

In the area between Las Palmas and Micos, some sharp anticlinial structures occur, in one of which several wells have been drilled (San Pedro), with a small production of light oil (45° Bé.). This, no doubt, will be further prospected; but as the production per well will probably continue small, and as operating is very expensive, it will probably not pay to go after such risky prospects when the low-gravity oils are so plentiful. The cost of producing this higher-grade oil will make it more profitable to import such refined products as cannot be made from the lower-gravity oils of the eastern fields.

The Topila district will probably be developed into an area where the salt-water problem will not be so serious; and probably such areas as Caracol and Los Esteros will be finally developed into better producers, although the first wells have been disappointing. There is then a great area north of Tampico, along the coastal plain between the Panuco River and Texas, in the part of which south of Soto la Marina (San Jose de las Rusias), at any rate, prevailing conditions are similar to those in the southern region. The Dutch Shell interests have large holdings in this northern area. There are many seepages and igneous intrusions in a part of this region.

The district between Las Palmas and Ebano and thence south to Tantoyuca, has not been prospected, owing largely to transportation difficulties. Before the present political difficulties, a railway had been begun, to connect by a short route between Tampico and the city of Mexico. When completed, this will open up the area mentioned. While in general this area does not seem to be so favorable as that east of Otontepec, yet not so much is known of its geology and structure, and it is probable that good pools will be developed, possibly of lighter oil than that of the present fields.

In the extreme eastern belt, along the coast between Tampico and Tuxpam, owing to the fact that the full thickness of the upper Tertiary formations must be passed through, together with considerable thickness of still more recent deposits, drilling at many points will have to be very deep before encountering either the San Felipe beds or the Tamasopa lime. At Tanguijo the upper Tertiary beds are at least 1,200 ft. thick, and a well 4,000 ft. deep had not passed the San Felipe. Moreover, the frequent deficiency of surface indications through the relative scarcity of seepages; the lack of intrusives appearing at the surface; and the possibility of seepages having travelled a considerable distance through the extra thickness of upper formations before appearing at the surface; —are all factors making it difficult to locate wells with the same chance of success as elsewhere. Seepages here are not so significant of accumulations or strongly fractured zones directly below, as in regions where the upper beds are thinner. It is also argued that so far down the dip to the eastward there is more danger of encountering salt water. This is not so dangerous as would at first appear; for oil would be caught in the upper part of fractures, while the water would encroach along the lower portion, and thus "spill" around the entrapped oil, much as happens in the case of encroaching water passing and retaining the oil in the crest of a sharp anticline. Such wells might possibly be shorter lived, but this is theoretical and can only be established by the drill.

V. Probable Future of the Mexican Fields

1. *Evidence of Producing Wells*

The first important producing well in the Mexican fields was the Mexican Petroleum Co.'s Pez No. 1 in the Ebano district. This well has been producing since 1904, and still continues to flow with little diminution at more than 1,000 bbl. per day.

The famous Juan Casiano No. 7 has been producing continuously since September, 1910, and is now making 700,000 bbl. per month, and has approximately 40,000,000 bbl. to its credit.

Nor are these isolated cases. The Pearson No. 4 well at Potrero del Llano has done at least as well as Casiano No. 7 for about the same length of time, and the older wells at Panuco are still good producers in the most closely drilled field of all. While the Chila-Salinas and Topila fields have suffered from the invasion of salt water in the past, yet recent development work indicates that adjoining areas may be better in this respect. The Dos Bocas gusher blew a great crater in the ground, and after producing 200,000 bbl. of oil per day for 57 days, went to salt water. Large quantities of asphalt are still thrown out with the salt water and gas from this hole, and there is reason to believe that if this well had not got beyond control it would have held back the salt water until its oil production would have been much greater in the aggregate than the actual amount expelled during its short life. As already observed, the larger wells in a number of districts are inclined to increase their production after being allowed to flow wide open for a short time. All the evidence of past wells points to the long life and high productivity of most of the districts already tested, and the probability of many more being eventually opened up.

2. *General Summary*

It is believed that Mexico is potentially the second greatest oil-producing country in the world today. The reasons for this belief are summarized below, as well as the causes which will tend to hold back for some time the realization of the possible maximum production:

1. A large number of widely separated areas have been tested and have developed uniformly large wells.

2. With the exception of a few initial wells, none of these areas has been fully developed by the drill. There is thus a considerable acreage undeveloped in already proven territory.

3. Numerous promising localities remain untested in the present fields.

4. Large districts yet untested, such as those north of Tampico and west and northwest of Otontepec offer surface evidence of the same characteristics as the proven fields.

5. The production per well has been large, and the decline slow in most fields.

6. The underground accumulations in fractured zones are capable of yielding a much larger percentage of their petroleum content than is a sand body.

7. More efficient methods of production result from the usual control of each producing district by one large company.

If such districts were located in the United States, they would soon be drilled up to a production such that the price would break to a very low point; but in Mexico this maximum production will doubtless be reached much more slowly, because of: (1) internal political difficulties; (2) delay in opening up isolated districts by railroads and pipe lines; (3) the many large tracts of land in most districts, the expense of handling which keeps out small operators, who would over-drill; (4) the high expense of operating, which keeps out a host of adventurers; (5) the lack of tankers for transportation; (6) the difficulty, by reason of expensive production and transportation, of competing in the United States market with Gulf Coast and California fuel oils; and (7) the present over-production of high-grade oil in the United States, which will delay the adaptation to Mexican oils of the new processes for refining heavy oils. These economic conditions account for the fact that Mexico, from wells having a present daily capacity of from 330,000 to 500,000 bbl., is producing only 70,000 bbl. a day, and that no active endeavors adequate to this discrepancy are now in progress, for the liberation and conveyance to the sea-coast, of the enormous amount of oil thus shut in.

TRANSACTIONS OF THE AMERICAN INSTITUTE OF MINING ENGINEERS
[SUBJECT TO REVISION]

DISCUSSION OF THIS PAPER IS INVITED. It should preferably be presented in person at the San Francisco meeting, September, 1915, when an abstract of the paper will be read. If this is impossible, then discussion in writing may be sent to the Editor, American Institute of Mining Engineers, 29 West 39th Street, New York, N. Y., for presentation by the Secretary or other representative of its author. Unless special arrangement is made, the discussion of this paper will close Nov. 1, 1915. Any discussion offered thereafter should preferably be in the form of a new paper.

The Cost of Maintaining Production in California Oil Fields

BY M. E. LOMBARDI, BERKELEY, CAL.

(San Francisco Meeting, September, 1915)

THE cost of maintaining the production of an operating oil company is one of the most important, as well as one of the most difficult to estimate, of the various items which go to make up the total cost of producing oil. In the opinion of many operators the cost of drilling and developing new wells, whose production takes the place of the loss in production through decreased yield of the old wells, is considered an operating expense, for the reason that the total income-producing power of the property is not increased. The cost of these new wells may be added directly to production expense, or it may be charged to capital account, and this account depreciated to the same extent. In either case, after an oil company is on a satisfactory income-producing basis, the money for drilling new wells to maintain production must be taken from the income of the company.

Three factors determine the cost of maintaining production: First, the rate of decline in yield of the wells; second, the cost of drilling and equipping new wells in terms of their production; third, the decrease in initial production of new wells as the property is drilled and gas pressure taken off the sands, etc.

In order to check theoretical figures for the cost of maintaining production the writer, with the assistance of Reed Bush and Fred Tough, has compiled the following figures. It is necessary first of all to choose a period of time during which production for the field or territory studied remains fairly constant. The longer the period of time over which this condition prevails, the better. It is also necessary to eliminate or allow for any extraneous condition which may have affected the production returns, such as the shutting down of a large number of wells, the bringing in of a well of exceptionally large yield, the opening up of virgin territory where the flush yield is greater than will be had from new wells in after years, etc.

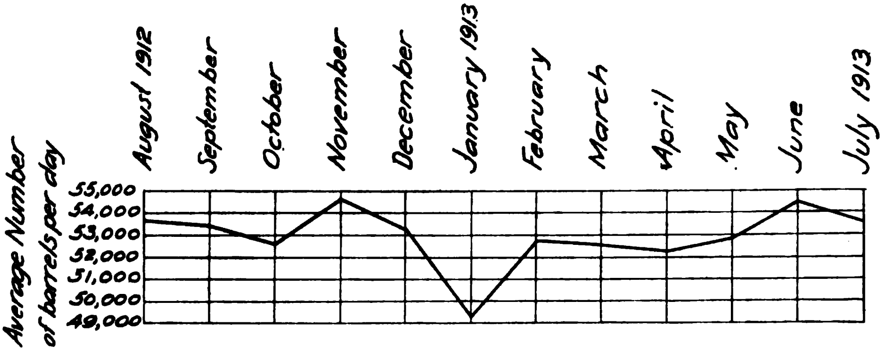
All these conditions are exemplified most simply in the case of the Coalinga field, and perhaps these figures are more truly prophetic than the two other cases given. In applying these figures to the future it is well to assume that the average yield of new wells will constantly decrease, since the flush-production period is past for most of the territory. On the other hand, the cost of drilling will be constantly less, owing to improved methods and particularly to the increasing use of the rotary drill. Although a great many estimations must be made, the results check closely enough to be interesting and possibly of value.

The data used are from the records of the Producers' Agency, the *Standard Oil Bulletin*, and various private sources.

It is to be noted that the figures for a number of wells at the beginning of the periods include producing wells only, but that all wells drilled during the period, whether profitable or not, are included in the calculations, provided they are within the accepted limits of the field—not pure wild cats.

Coalinga Field

In the Coalinga field the period from Aug. 1, 1912 to July 31, 1913 was chosen. The production curve is shown in Fig. 1. It will be noted that the production during these 12 months was remarkably uniform, the greatest variation from the average being about 8 per cent. and the average variation on a monthly basis being about 2.8 per cent. During this period 19,313,000 bbl. of oil were produced and 78 new wells were completed. The number of wells suspended during the period varied from 176 to 214 out of a total of 843 wells at the beginning. These wells were "off" from natural causes, and were not to any extent intentionally shut in. The greatest variation from the average number of wells suspended was only 8.6 per cent., so that this factor of wells shut in is not of great importance in the results.



Total production for period, in barrels	19,313,000
Wells completed for period	78
Estimated total cost of these wells	$2,176,573
Cost of wells per barrel produced	$0.1126
Number of wells at beginning of period	843
Per cent. of new wells to maintain production per year	9.33

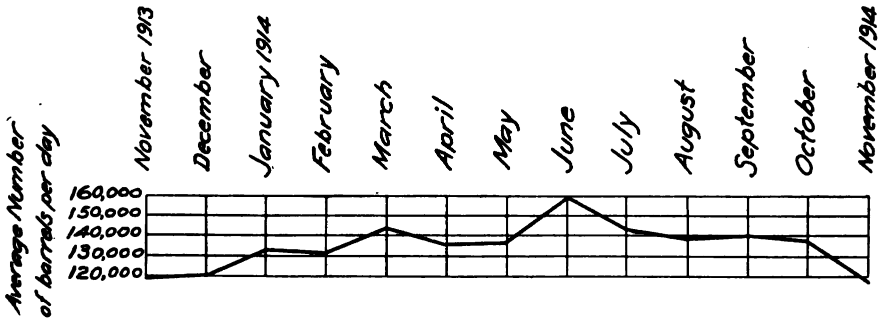
FIG. 1.—CURVE OF TOTAL PRODUCTION OF CRUDE OIL IN COALINGA FIELD, CALIFORNIA, FOR THE PERIOD AUGUST, 1912, TO AUGUST, 1913—12 MONTHS.

The main facts are, that at the beginning of this period 843 wells were producing, and that, in order to maintain the production of these wells, 78 new wells had to be drilled. To arrive at the cost of these 78 wells, representing the cost of maintaining production, the new wells were classified in four groups. The actual cost of 95 wells in this field (also divided into groups of similar characteristics) was accurately known to the writer. These average cost figures applied to the 78 wells in question showed their cost to be approximately $2,176,573. This cost includes derrick and rig, well, flow-tanks and flumes, gas engine or other local pumping equipment, but not such items as storage facilities, pipe lines, main power plants, etc. If this figure is divided by the total number of barrels produced during the period, we find that 11.26 c. per barrel of oil produced is the cost of maintaining production in this field.

It is also interesting to note that the number of wells necessary to maintain production for one year was in this case 9.23 per cent. of the total number of wells producing at the beginning of the period.

Sunset-Midway Field

In the Sunset-Midway field the period from November, 1913 through November, 1914, was chosen. Fig. 2 shows that the production was more than maintained, that is, that it was higher at all times during the period than at either end. The Lakeview No. 2 gusher came in in May, 1914 with a production of about 25,000 bbl. However, the greatest variation of the average production was not more than 10.5 per cent. During this period seven gas wells were drilled. These are not included in the figures given, as it is assumed that they were drilled to produce gas only, and no attempt made to produce oil.

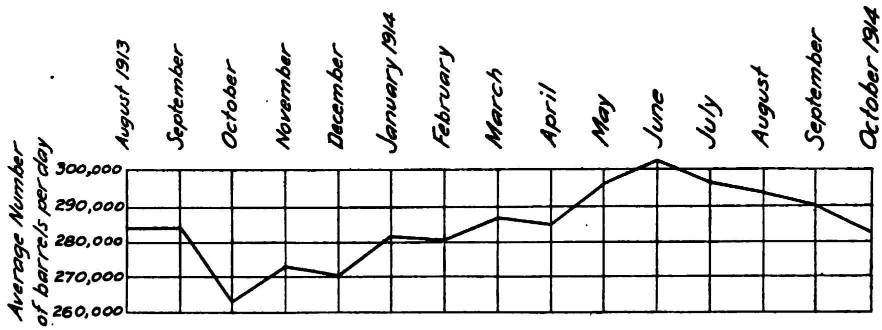


Total production for period, in barrels	52,799,118
Wells completed for period	189
Estimated total cost of these wells	$4,966,270
Cost of wells per barrel produced	$0.096
Number of wells at beginning of period	1,177
Per cent. of new wells to maintain production per year	14.8

FIG. 2.—CURVE OF TOTAL PRODUCTION OF CRUDE OIL IN SUNSET-MIDWAY FIELD, CALIFORNIA, FOR THE PERIOD NOVEMBER, 1913, TO DECEMBER, 1914—13 MONTHS.

During this period, a wide-spread agitation occurred in favor of limiting production, and a good many wells were closed in. The probable production of these closed-in wells was estimated as closely as possible and added to the actual production of the field, in order that this intentional decrease should not affect the results, which are of necessity based on normal flow from each well.

It will be noted that the total oil produced was 52,799,118 bbl. including estimated shut-in production, and that 189 wells were drilled during the period. The cost of these 189 new wells was estimated in the same manner as for the Coalinga field. The costs of 95 wells in various parts of the Sunset-Midway field, being accurately known to the writer, were applied to the 189 wells with due regard to the location of each well, etc., the same items being included as in the case of Coalinga. The resulting estimate is $4,966,270 for maintaining production during the period, which amounts to 9.6 c. per barrel of oil produced.



Total Production for period, in barrels	130,067,732
Wells completed for period	562
Estimated total cost of these wells	$13,559,000
Cost of wells per barrel produced	$0.104
Number of wells at beginning of period	6,069
Per cent. of new wells to maintain production per year	8.22

FIG. 3.—CURVE OF TOTAL PRODUCTION OE CRUDE OIL IN CALIFORNIA, FOR THE PERIOD AUGUST, 1913, TO NOVEMBER, 1914–15 MONTHS.

The proportion of new wells necessary to maintain production for one year in this case is 14.8 per cent., which is a considerably higher figure than for Coalinga, although the cost per barrel produced is less. This is explained to some extent by the fact that a larger number of new wells in the Sunset-Midway field were drilled in shallow and "easy" territory in the northwestern part of the field, with a correspondingly low cost per well.

All California Fields

A study was next made based on the production of the whole State of California for a period of 15 months beginning August, 1913. During this period, as shown in Fig. 3, the greatest variation from the average production was only 7.5 per cent., which is remarkably small considering the period.

At the beginning, approximately 6,069 wells were producing and during the period approximately 562 wells were brought in, costing approximately $13,559,000. This figure for the cost of new wells is not as accurate as desired, because the writer is not familiar with the cost of wells in some of the smaller fields. Average cost figures taken from over 250 wells in four fields were used. During the period approximately 130,067,730 bbl. of oil were produced, and dividing the cost of the new wells by this production, we find that the cost of maintaining production was 10.4 c. per barrel of oil produced. This differs from the Coalinga figure by only 0.86 c. per barrel, and from the Sunset-Midway figure by 0.8 c. and is very close to a mean between the two. It is also interesting to note that the number of wells necessary to maintain production over the whole State for one year was 8.22 per cent. of the wells producing at the beginning of the year, which compares favorably with the figure of 9.23 per cent. for Coalinga.

Cost per Barrel of Daily Production

It is also interesting to apply the data given above to estimate the cost of maintaining production in terms of the average daily production. For example: A company operating average leases in various fields, having a daily production of several thousand barrels, may wish to estimate from a general average how much money will have to be appropriated per year for new drilling to maintain its production.

In the following calculations the average daily production for the whole period is assumed as the production at the beginning of the period. In Coalinga the daily production may be thus assumed as 52,912 bbl. at the beginning of the period illustrated above. It, therefore, cost $40.38 per barrel of daily production to maintain production during the ensuing year.

Likewise for the Sunset-Midway field, the figure is $33.70, and, for all fields in California, $38.

It must be borne in mind that, in this case, as in the case of the cost per barrel produced, these figures show the cost for wells and their immediate appurtenances only: The cost of extension of pipe lines, new permanent power plants, camps, water systems, etc., must be added. A fair figure for these is between 12 and 20 per cent. of the cost of wells.

Depreciation

If we apply the cost figures used above to all wells producing at the beginning of the various periods shown, the apparent depreciation in earning power of investment during the ensuing year is shown by the ratio of new wells to old. In Coalinga it is 9.23 per cent., in the Sunset-Midway field 14.8 per cent., and for all California fields 8.22 per cent.

It must be borne in mind that these figures apply only when considering wells producing at the beginning of the period. That is, no account in investment is taken of past failures or abandoned wells. Also that the figures show an average depreciation for wells in all stages of life. It is well known that new wells fall off in production as much as 50 per cent. in their first year, whereas wells 5 or 6 years old decrease very slightly in comparison.

It must also be borne in mind that many of the new wells which maintained the production during the period illustrated were in practically virgin territory in the extension of the known fields, or in new fields, and that their initial production was correspondingly high. This condition will not exist long, and it will be safe to discount the results given above for future use.

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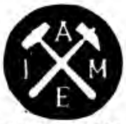

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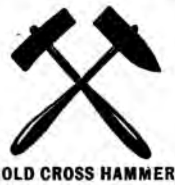

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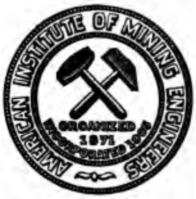

SEAL FOB

THE AMERICAN INSTITUTE OF MINING ENGINEERS has adopted no official badge, the possession of which is proof of membership. Numerous designs, however, have been executed for temporary use, and of these three here shown have proved so convenient and attractive, that many members continue to wear them as personal ornaments. To this there is no objection; and these badges will be sent postpaid to Members and Associates on the receipt of the price by New York check, or Postal Money Order, in favor of the Institute, as follows:

Blue Enameled Pin, Silver, \$0.75; gold, \$2.00; White Enameled Button for officers, gold, \$2.00; Cross-Hammers, gold, \$2.00; Seal Watch-Fob, silver, \$2.25; silver-gilt, \$2.75; gold, (14 kt.) \$14.00.

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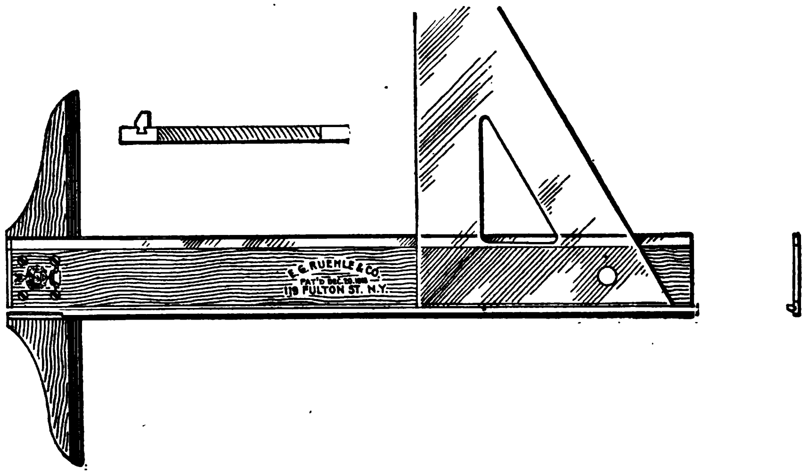

The extension strip of this new implement permits the placing of the triangle right on top of the wet lines without touching and thus blurring them—leaving them intact—and work can go on uninterruptedly while "inking in."

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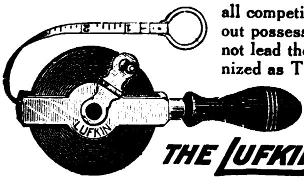

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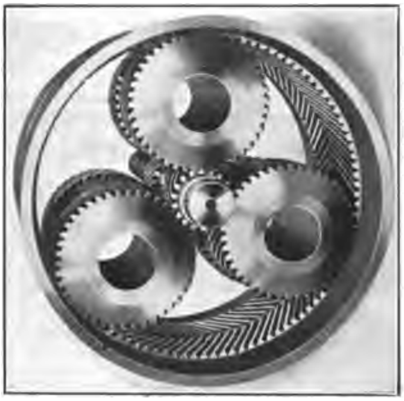
By use of the Turbo-Gear in electric motor driving it is possible to secure a slow speed without the use of sprockets and chains, pulleys and belts, extra shafting and bearings, etc., which require larger floor space and must be enclosed with guards to prevent accidents.

The manufacturers speak of the Turbo-Gear as a speed transformer because of its being unlike gearing. It is said to be so quiet and smooth in operation that it is not comparable to gearing in the ordinary sense of the word. It is said to be equal in efficiency to the best herringbone gear made, but is much more compact.

The Turbo-Gear consists of a large internal double helical gear made of a special analysis open-hearth steel forging, heat treated to increase its ductility and to insure uniform hardness. A double helical pinion cut integral with the high-speed shaft, made of Halcomb electric furnace chrome-vanadium steel, heat treated to an elastic limit of 210,000 lb. per square inch and of proper hardness to minimize wear. Intermediate double helical gears made of Lumen manganese bronze are mounted on hardened and ground forged-steel shafts secured to the cast-steel slow-speed member by means of taper fit and Woodruff keys.



This slow-speed member, to which is secured the slow-speed shaft, is mounted on two heavy-duty S. K. F. ball bearings, one on each side of the gears, and supported directly by the heavy housing. Thus it will be seen that the slow-speed member and shaft carrying the intermediate gears, and the high-speed shaft and pinion, are independent of each other for support and that each is supported directly by the housing.

The high-speed shaft and pinion as well as the slow-speed member carrying the intermediate gears are accurately ground all over and are carefully tested for balance before being assembled.

The housing is made of tough gray cast iron, horizontally split in accordance with most approved practice, affording perfect accessibility to all internal parts. It is extremely heavy and well ribbed to provide a rigid support for the gear members, which is essential to efficient and

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quiet operation. The high- and low-speed bearings are provided with caps effectually protecting them from dust.

The high-speed shaft has a central passage through which the oil is pumped and a continuous stream of oil is sprayed on the gears through radial passages in the pinion.

The high-speed bearings besides having forced feed lubrication are provided with oil rings and a good-sized oil reservoir for emergency use.

The superfluous oil from the high-speed bearings is collected by a centrifugal oil ring and forced through the hollow shafts carrying the intermediate gears, flushing their bearings.

The oil after lubricating the bearings and gears is immediately drained to the main oil reservoir in the base of the housing: here it is strained, cooled, returned to the pump and used over again.

It will thus be seen that the gears do not run in oil, which causes a considerable back pressure at high speeds with corresponding loss in efficiency.

It is said that tests have shown efficiencies as high as 98 per cent. and 99 per cent. at full load.

Further details will be furnished on application.

Exhibit of the Lagonda Manufacturing Co. at the Panama-Pacific Exposition

The exhibit at the Panama-Pacific Exposition of the Lagonda Manufacturing Co., of Springfield, Ohio, consists of an interesting display and

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effects big savings in Tramming Costs at the mines of the Portland Gold Mining Co. Our order for two duplicate units is the result.

Locomotive operates over 18 inch gauge track a distance of 800 feet. On each charge of battery it makes 24 round trips handling a train of 10 cars on each trip. Has been in continuous operation for nearly a year, working two shifts of 8 hours each.

Send for Bulletin No. 107-2 which gives further particulars, and tells about the cost-reducing and labor-saving advantages of the Jeffrey Storage Battery Locomotive

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From the steel furnace to the finished rope every process of manufacture is conducted so as to result in a **serviceable** rope.

John A. Roebling's Sons Co.

Trenton, N. J.

illustration of the operation of their boiler-tube cleaners, automatic cut-off valves, multiple strainers, and grease extractors.

Referring to the illustration of the Exhibit, to the left are the high-power direct motor-driven tube cleaners for boilers, condensers and economizers, suspended and inserted in condenser tubes and the tubes of a B. & W. manifold, just as they are in actual use. Directly behind these cleaners can be seen the handwheels of the Types A and B outside and internal dashpot automatic cut-off valves, and the Type C or internal dashpot non-return valve.

At the back of the exhibit and to the extreme left is an exhibit of the Lagonda Stirling boiler-cleaning device, by which the entire operation of cleaning is conducted from without the drum. The device is supported in an oval frame, representing the manhole, and has the cleaner and rubber operating hose inserted, just as in operation. Immediately to the right of this and to the left of the booth are shown the electric and air or steam motor-driven reseating machines, for reseating and grinding fitting surfaces and cleaning scale and soot from caps and nuts of B. & W. and similar type boilers.

To the right is a stand containing a full display of all types of water, air and steam turbine driven cleaners for water and fire tube boilers, condensers and other tubular apparatus, also cleaners for curved-tube boilers.

On a second stand directly behind the turbine cleaners are exhibited Lagonda tube cutters and a sectional model of the Type A automatic cut-off valve, showing the construction and operation of the valve and external dashpot. Directly behind this is the new Lagonda grease extractor and a three-basket multiple strainer. The grease extractor operates on the principle of the Lagonda multiple-basket water strainer.

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Note the horizontally split casing, allowing quick, easy access to all working parts.

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¶ It can be made almost negligible.

¶ Or it can be made excessive.

¶ Sometimes it's the fault of the hoist.

¶ But isn't it often the fault of *contributing* conditions, or the operator.

¶ Take VULCAN hoists, for instance. We take the utmost pains in the making. We test and analyze every casting, every forging; our own specifications are more rigid than yours. There's not a thing left undone to make maintenance *low*.

¶ And if it is not low, it's generally traceable to outside conditions that can be remedied easily.

¶ Keep your hoist clean, well-oiled and greased and test all the nuts and bolts now and then, and give the bearing wedges a look.

¶ Then you'll see maintenance worries disappear.

VULCAN IRON WORKS
1744 Main Street
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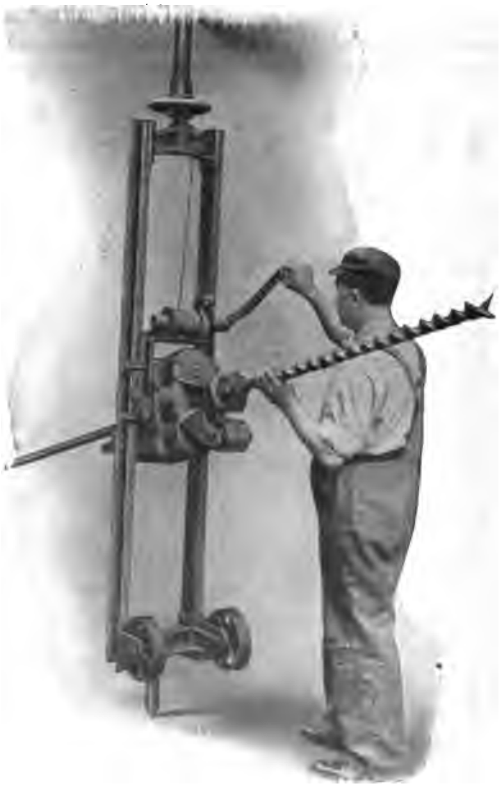
The demonstration of the working of the multiple strainer is especially interesting. A three-basket type strainer has been constructed with a glass panel at one end, so that the flow of water through the strainer and the accumulation of solid matter in the basket can be seen. In operation the small two-basket strainer body shown at the left is used for inserting sticks, leaves and other such refuse as is found in plant-supply water, then the water is turned on and allowed to flow through the strainer. All the solid matter passing into the strainer is caught in the strainer baskets. It is then demonstrated how the strainer baskets can be individually drawn out of commission, removed and cleaned without interrupting the operation of the other baskets or the flow of water.

Charles C. Moore & Co., Western representatives of the Lagonda Company, have charge of the exhibit.

New Type of Electric Power Drill

The accompanying illustration shows a new type of powerful coal drill recently developed and placed on the market by the Jeffrey Manufacturing Co., and known as their A-5-C electric power drill.



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The motor and drill mechanism proper are mounted on trunnions in a carriage, which may be raised or lowered in a vertical channel-iron frame by means of a $\frac{3}{8}$-in. steel rope. One end of this rope is fastened to the top yoke of the frame and the other end to a drum on the carriage. This drum is rotated by a handle, as shown, to wind up the rope and to raise the carriage. A ratchet and pawl are provided on the drum to prevent the carriage from dropping when the handle is released.

When it is desired to move the drill from one place to another, the carriage carrying the motor and mechanism is lowered to the bottom of the frame and the drill rolled along on the wheels provided on the bottom cross yoke, in a manner similar to an ordinary hand freight truck.

The motor of this drill is rated at 3 hp. The total weight of the drill is approximately 300 lb., while it is made sufficiently rugged and powerful to drill through any material against which the auger will stand up.

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Further particulars concerning this drill, also latest catalogues and bulletins on the Jeffrey complete line of coal-mine and tipple machinery, may be obtained by writing to the Jeffrey Manufacturing Co., 902 North Fourth Street, Columbus, Ohio, or to any of their branch offices.

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A new method of recording the cost of various conveyor belts in operation has been worked out by the B. F. Goodrich Co., of Akron, Ohio, in the interest of all belt users.

Ordinarily accurate costs are rather difficult to determine and many operators have not the time to work out an adequate system for making computations. The use of various belts, however, is an important factor and many a dollar can be saved when you know where to cut down the tonnage cost.

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Output of mine in one hour	320 tons from 2000 ft. level
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Maximum Hoisting Speed	3000 ft. per minute
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ALPHABETICAL LIST OF ADVERTISERS

PAGE
Albany Lubricating Co. . . . 5
Allis-Chalmers Manufacturing Co. . . . 16
Beer, Sondheimer & Co. . . . 21
A. S. Cameron Steam Pump Works . . . 11
Crocker-Wheeler Co. . . . 16
Denver Fire Clay Co. . . . 16
Denver Rock Drill Co. . . . 16
Dorr Cyanide Machinery Co. . . . 5
Edison Storage Battery Co. . . . Inside Back Cover
Ensign Bickford Co. . . . 13
General Electric Co. . . . Outside Back Cover
Goodman Manufacturing Co. . . . 17
B. F. Goodrich Co. . . . Inside Front Cover
W. & L. E. Gurley . . . 1
Harbison-Walker Refractories Co. . . . 18
Illinois Zinc Co. . . . 18
Ingersoll-Rand Co. . . . 13
Jeffrey Manufacturing Co. . . . 9
Lead Lined Iron Pipe Co. . . . 18
A. Leschen & Sons Rope Co. . . . 19
Lufkin Rule Co. . . . 5
Mashek Engineering Co. . . . 19
Robins Conveying Belt Co. . . . 19
John A. Roebling's Sons Co. . . . 9
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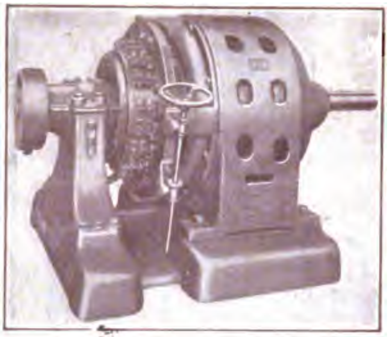

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